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Toronto Meeting of the American Electrochemical Society.

The report, printed elsewhere in this issue, of the Toronto meeting of the American Electrochemical Society held from September 21 to 23, will be found so complete that little could be added to the details of this highly enjoyable and successful convention. But we may use this opportunity to say a few words on some broader aspects of the remarkable development of this Society which is not yet ten years old.

Firstly, the Toronto meeting—the second one held on Canadian soil—confirmed what scrutiny of the membership list and of the papers published in the Transactions has long indicated: that in one respect at least the Society has already outgrown its name. It is no longer a national, it is an international society. In view of the unbounded hospitality which the visitors enjoyed at Toronto and the splendid arrangements which had been made by the local committee with Dr. W. Lash Miller as chairman and Mr. W. P. Cohoe as secretary, it goes without saying that everyone felt very much at home in Canada. And the coincidence of the first convention day being the day of the Canadian anti-reciprocity election only served to emphasize more strongly that in the interchange of ideas there can be no tariff. In the realm of exchange of ideas reciprocity expands freely by God-made laws.

Secondly, the glorious resurrection of Section Q with their clever and funny electric-furnace farce, presented at the smoker, deserves a word of notice. A keen observer who has attended many meetings of societies here and abroad has remarked that it is at the meetings of the American Electrochemical Society that "the most human crowd" is found. Nor is this to be wondered at. The Society is devoted to a borderland of sciences and industries and much of its work is pioneer work in virgin fields of science and engineering. We need only think of Bret Harte's stories to see how pioneer work of any kind brings out the human element, how it makes men respect each other for their accomplishments and love each other for their shortcomings, how it breeds a sense of humor. This was the spirit in which Section Q was born and this is the reason why the Electrochemical Society could not be without it. The fun of the farce presented at Toronto was so clever and good-natured that nobody could be hurt by the hits made in the fanciful play. Goethe's maxim of life of "Tages Arbeit, Abends Gäste, saure Wochen, frohe Feste," was followed in that long to be remembered smoker after the two heavy professional sessions of the first day.

Thirdly, for a Society with such a record as the American Electrochemical Society the danger would generally be great to rest on its laurels. But in this case the spirit of youthful unrest inherent in all electrochemists would not permit a policy of "let well enough alone." This is forcibly brought out in the report of the special committee appointed to consider and report "whether a higher standard of selection of papers read

at the meetings or published in the Transactions should be observed" and in the unanimous adoption of the committee's recommendations by the Board of Directors. These recommendations are now to be worked out to be carried into effect in due course of time. They involve no criticism of the splendid work done in the past, but recognize distinct possibilities of doing better in the future. As long as such a self-conscious and critical spirit is alive in those who are intrusted with the Society's management there can be no doubt that the career of the Society must continue to be one of ever-increasing usefulness.

The Eighth International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry—the first one to be held in this country—is now only one year off. The time has, therefore, come for everyone to start energetically to do his share to make the Congress a success worthy of this country and American chemistry.

The Congress has been organized on the principle of broadest chemical democracy. As the Acting President, Dr. Wm. H. Nichols, remarked in one of his early speeches: "Every chemist in the United States must feel that he is part of the Congress from the very creation of the organizing committee down through all the details of management to the very close of the Congress."

The different sections have been organized and are ready to begin their systematic work. But every single chemist in the United States himself, directly or indirectly through his professional, business or educational affiliations, has a personal share of responsibility in the success of the Congress.

The success of the Congress will depend essentially on the contents of its printed Transactions. Thus the securing of the papers becomes the most important task of the near future. Tentative rules and regulations for the papers have been formulated. They are printed in full on pages 510 and 511 of this issue. Since they will be adopted in their final form in December, it is necessary that any criticisms or suggestions should be made to the Secretary of the Congress before December.

But aside from this more formal matter, the all-important point must be kept continually in mind that everything be done to secure for the Congress any worthy paper and thus to make the Transactions a truly representative document

Transfer of the Ray Smelter.

The Ray Consolidated Copper Company, Arizona, was planned as a complete copper mining, concentrating and smelting concern, with full equipment for carrying on these several operations. Recently, however, the smelting end of the project has been abandoned before its completion, and the plant has been sold to the American Smelting & Refining Company. Apparently the Ray company intends to stick close to mining and concentration, in which fields its promoters have gained signal success; while the Smelting company takes advantage of an excellent opportunity to extend its smelting operations and gain a foothold in Arizona.

The arrangement should be advantageous to both companies, each of which will operate in its chosen and successful field.

The Ray company has assurance that its concentrates will be treated by a friendly company practically as cheaply as in its own plant, and at the same time it is relieved of the arduous work of conducting a large smelting industry. The smelting company, on the other hand, enters into competition in one of the largest copper-producing States in the Union, and in a district where almost 150,000,000 tons of low grade copper ore are developed.

The promoters of the Ray began their immense operations in the Utah Copper Company, later developing the Ray in Arizona and the Chino in New Mexico. They also have been interested in Nevada Consolidated. Almost from the beginning they have had close working relations with the Guggenheims, and the recent transfer of the Ray smelter to the latter is a logical if unexpected move.

Some Recent Crushing and Grinding Tests.

Evidence is accumulating on the subject of crushing and grinding which forces a modification of some views formerly held, and the results obtained in some recent tests must be accepted regardless of stubborn prejudice or nice theory. In the Synopsis of Current Literature in another part of this issue reference is made to the work of ball mills in Australia and the excellent performance of these machines in dry crushing. Three-inch ore is crushed to pass approximately 30-mesh screen, with a power consumption of about 0.5 hp per ton. The quantity of minus 150-mesh material produced is not as great as with other pulverizers and the product is sharp and granular even in the finer sizes.

This reference to the ball mill recalls the belief which is finding acceptance among many engineers that the ball mill and tube mill in tandem will be found a most efficient system of wet grinding. This belief is substantiated in a large measure by the recent tests of the so-called Giesecke ball-tube mill in South Africa. This machine is a ball mill and tube mill combined, with compartments for coarse crushing and fine grinding, and a suitable connection for passing the product of the former to the latter. It has a total length of 24 ft., of which 5¾ ft. at the feed end is the ball mill compartment with a diameter of 7 ft. 6½ in., and the balance is the tube mill with a diameter of 5 ft. 10 in. Steel balls of various sizes ranging from 1¼ in. to 4 in. are used as the crushing medium. The larger sizes are used in the ball end and the small in the tube. Their total weight is 23 tons. The liners are 2-in. iron plates with one end bolted to the shell and the other projecting inward about 9 in. The entire mill weighs 68 tons without load, runs at 24 r.p.m. and requires about 235 hp when running. Its capacity is 360 tons per day of 7-in. material, which is reduced so that 80 per cent will pass a 200-mesh screen.

Here is a departure in grinding machines, with preliminary results that are very attractive. If the cost of operation and maintenance should prove to be no greater than that of stamps and tubes a marked saving is possible in cost of installation as compared with the latter combination. And even if the Giesecke mill in the form described should, for various reasons, prove an unwieldy combination, there still exists the probability that separate ball and tube mills may prove an efficient combination.

One feature in particular which contributes to the capacity

of the Giesecke mill is the small quantity of water used in grinding—just sufficient to make the pulp flow. This insures an efficient grinding action of the balls approaching that of the dry ball mill, where no fluid other than air can tend to wash the particles of ore out of the path of the rolling and tumbling balls.

The Giesecke mill or the separate ball and tube mill combination gives stage crushing in a compact apparatus. Judging from the success attained by the ball mill and the tube mill when used separately for specific purposes, their combination should be more thoroughly investigated, and the tests made in South Africa will serve to draw more attention to their possibilities.

Of a different nature, but relating to the subject of crushing, were the tests on rolls and Chilean mills as grinders of disseminated copper ore at the Miami mill in Arizona. In the construction of this plant one section was equipped with rolls and the others with Chilean mills. The tests have been completed to the marked advantage of the Chilean mills, which, although noted as producers of slime, did more efficient work in liberating the copper mineral from gangue than the rolls, which gave a more granular product. This test adds to the prestige of the Chilean mill in copper concentration in the West and gives it a position which it holds apparently through merit alone.

Wages in the Steel Industry.

In the past four years the steel industry has come to occupy a very unfortunate economical position. Its selling prices were admittedly not inordinately high in 1907, although they were entirely satisfactory. From the average prices of that year to the present time the decline has been fully \$10 a gross ton as the average of all products outside of standard sections of steel rails, which are unchanged in price but are subject to much more rigid specifications and to a higher manufacturing cost. The general average level of commodity prices in the United States has not declined materially, if at all, since 1907, so that the steel industry stands at a very great disadvantage, in having suffered a reduction of \$10 a ton in its selling prices, equal to between 20 and 25 per cent, computed upon material in the finished rolled form.

The average wage rate in the steel industry is higher than in 1907, through the general advance which was made in the spring of last year. The average increase is not large, but the fact that the change in wage rates has been in this direction places strong emphasis upon the peculiar economic position resulting from the great reduction in selling prices.

The last vestige of artificial control of prices in the steel industry appears to have vanished, if the maintenance of the \$28 price on standard section rails be disregarded. It may be due to a survival of the co-operative idea or may be due to a refusal of the railroads to entertain lower prices. The railroads realize fully that the price is not high for the quality of rail they desire, and that a period of open competition in rail prices would make it much more difficult to secure the improvements in quality which they desire. Thus we may say that there is no longer any artificial control in steel prices through joint action or tacit understanding among the steel producers. The course of the markets in the past three months

has clearly betokened conditions of open competition. Declines have not been clear cut, as have been those which occurred during the long period of a controlled market, but have been of the nature of a gradual sagging, with resistance on the part of sellers yielding to the necessity of making concessions in order to obtain individual orders. The downward trend in prices continues at this time and no indication is found in the markets that it will be arrested other than by prices reaching the cost level. Below that level they may possibly go, for most of the producers have large reserves and may become willing to make sacrifices in order to retain their trade. The "battle of the giants" which the daily newspaper paragraphers have delighted at times to predict is not among the possibilities, however, unless the total tonnage of demand should fall much below the present level, for the "giants" have not sufficient capacity to supply the existing demand. There are enough producers in a relatively poor position to be eliminated to leave a comfortable tonnage for those in better position.

In the circumstances outlined a general reduction in wages in the steel industry seems to be a certain prospect, and that not at a distant date but in the relatively near future. There is reason to believe that the subject has been thoughtfully discussed in the inner circles for several weeks past. In any circumstance a wage reduction is to be deplored, and would be a serious matter, but there seems to be no easy means of avoiding one.

The decline in steel prices has been so great that the average level of prices, outside of standard rails, is computed to be not more than about \$4 a ton higher than the lowest level ever recorded, the average prices in the years 1897 and 1898, when individual commodities fluctuated but little, being about \$4 a ton lower than the present level. Since then costs have increased in some respects and decreased in others. The wage rates in the iron and steel industry proper have greatly increased; the cost of ore mining has greatly increased, due to the necessity of seeking ore farther underground. Freight rates paid upon raw materials have materially increased, and practically all the commodities the industry buys are much higher. To offset these increases, which amount to very much more than \$4 per ton of finished product, the industry has introduced many economies in practice. Its wage rates have not, we may assume, increased materially more than rates in other industries, and not as much, probably, as in the building industry, and it is particularly unfortunate that no means seems to be afforded whereby savings can be effected in other directions.

The situation is simply this: The steel industry finds itself with much lower selling prices than formerly, and naturally turns to the wage rates it pays. If the general level of wages should come down, then it would be fairer that the industries which the steel industry patronizes, like railroading and the manufacture of its various supplies, should themselves reduce wages, seeing that the steel industry must pay them higher rates than formerly. The steel industry has effected many economies in its processes, and it is unfortunate that if it cannot keep the results as return upon its capital investment it cannot at least use them to avoid a reduction in wages. A fair adjustment based upon theory would call for economy to be practised at these outside points.

American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers will be held in Washington, D. C., from Dec. 20 to 22 (Wednesday to Friday).

A number of papers will be presented on the general subject of patents and on the manufacture and testing of explosives as well as on other chemical engineering subjects.

One day will probably be devoted to visits to chemical engineering plants in Baltimore and vicinity. Visits to laboratories and other points of interest in Washington will also be arranged for.

Latest Results in Electric Furnace-Smelting of Iron Ore.

The Trollhättan furnace is now working with round electrodes 550 mm. diameter, screwed into each other and thus entirely consumed. These screw-joints hold well, not giving any trouble by heating at the joints. While the electrode consumption of this furnace using square unjointed electrodes, from Nov. 15, 1910 to April 9, 1911, was 10.28 kilos per ton of pig iron, counting the lost butts, the consumption from Aug. 4 to Sept. 4, 1911 was 6.2 kilos, and it has now (middle of September), reached 5.5 kilos.

The furnace has now been running steadily since Aug. 4. The gas pipes have been made stronger, so that pressures of 0.3 to 0.4 atmosphere are now used under the vault. The ore is charged more towards the middle of the furnace shaft, and the percentage of carbon dioxide in the escaping gas has been increased to over 30 per cent.

The output of pig iron has been increased, the consumption of charcoal has decreased, and the output per horsepower year has reached the figure of 5.05 tons per kw year, or 3.7 tons per hp year, this being the figure reached in the week Sept. 3-9. The detailed figures for that week are as follows:

Ore	192.5 metric tons
Limestone	7.7 metric tons
Charcoal	44.2 metric tons
Electric energy used.....	228,000 kw-hours
Average power	1357 kw
Pig iron produced.....	131.4 tons
Slag	22.1 tons
Charcoal per ton pig iron.....	336 kg
Pig iron per kw-year.....	5.05 tons
Pig iron per hp-year.....	3.79 tons
Kw-hours per ton of pig iron.....	1736
Hp-hours per ton of pig iron.....	2315

The builders and operators of this furnace feel every reason to be satisfied with these splendid results, which fulfill all expectations.

At about the end of 1911, four similar large electric pig iron furnaces will be in commercial operation in Scandinavia, viz.:

- 1—2500 hp at Domnarfret, Sweden.
- 1—3500 hp at Tyssaa, Norway.
- 2—3000 hp at Hagfors, Sweden.

The Western Metallurgical Field.

San Juan District, Colorado.

The production of vanadium and uranium from this district is assuming the proportions of a considerable industry, and the prospect is that within the next year a great deal of attention will be given to mining and milling these ores. A great deal of unfounded and irresponsible talk is current regarding this district, especially regarding the alleged fabulous value of the ores and the probability of supporting a thriving radium industry. The facts seem to be that in San Miguel County there are commercial deposits of uranium and vanadium ores some of which are treated locally and others shipped abroad.

The only local treatment of vanadium ore in operation now is carried on at Newmire by the Primos Chemical Company. This company is operating under lease and bond, the mines and mill owned by the Vanadium Alloys Company. The latter concern worked its properties for some time, but met with difficulties in the mill work. These were in a measure due to the fact that the work was new and there were no guides to successful operation. In addition to this some of the equipment installed was unsuited to the work. The Primos Chemical Company has now been operating the mill for about two years and is in a fair way to make a success.

The ore treated is an impregnated sandstone the nature of which will be found more fully described in Mr. Bleecker's article elsewhere in this issue. The average grade of ore milled is not known exactly, but is not in excess of 1.5 per cent vanadium and probably lower. The details of treatment are not given out by the company, but the method probably does not differ greatly from that used by the Vanadium Alloys Company, the principal improvements being in the mechanical operation. The former company roasted its ore with salt in a McDougal roaster. This roaster as installed proved very unsatisfactory and gave rise to constant trouble. Difficulty was experienced in getting the heat evenly distributed to the various hearths and on one occasion a rabble arm broke by sticking in the fused mass and caused a serious explosion when the cooling water was injected onto the hot material. Although the same roaster is now in use, it is understood that the Primos company has remodeled it to use oil and that it gives fair satisfaction, although plainly not adapted to this kind of work.

The roasted ore is leached with water, getting a solution of sodium vanadate, and this is precipitated with ferrous sulphate. Various means of leaching have been tried at this mill, both with and without agitators, and the latter have been discarded. The vanadate of iron obtained by the precipitation with ferrous sulphate is shipped to Primos, Pa., where it is reduced to ferrovanadium.

The ores shipped abroad are rather higher in vanadium content, containing probably 5 per cent to 6 per cent. It has been a matter of surprise to some that this ore could stand the charge of transportation abroad, but one shipper is authority for the statement that the rate from Placerville, Cal., to Liverpool is 5 cents per ton less than from Placerville to Pittsburgh or common eastern points. The rate from Placerville to Pueblo, Col., is \$6 per ton; from Pueblo to Galveston, \$5, and from Galveston to Liverpool, \$2.50. Total, \$13.50. The rate to Pittsburgh is \$13.55.

Silverton District.

The mining and milling industry in the vicinity of Silverton is in a quiescent state, due to a number of causes. Briefly these are the complexity and low grade of the ores, and the fact that the processes of milling installed some years ago are not applicable to present conditions. Recently there has been a commendable tendency to investigate new and accessory processes and the prospect is fair for renewed activity in the near future.

One of the greatest difficulties is the separation of the complex ores into concentrated products of shipping grade. Iron, lead, copper and zinc occur together in many of the ores, and ordinary wet processes of concentration are not profitable. Recently there has been renewed interest in the possibilities of electrostatic separation as a means of reviving many dormant properties. Inquiry brought out the fact that many mills already built but not in operation would be benefited by adding electrostatic separation to their present process, enabling them to separate the copper and iron from the zinc and make the latter a profitable metal instead of a stumbling block. Probably a score of these mills could be so benefited immediately. Two solutions of the problem are in sight, viz., the installation of electrostatic machines in the individual mills or the establishment of a separate electrostatic separation plant to be supplied with zinc-iron-copper middlings from the various mills.

Undoubtedly the former plan would be more satisfactory to all concerned if the individual companies could raise the necessary money. The latter offers the advantages and disadvantages which any custom plant gives to a community, but mainly its success depends on the assurance of a constant and adequate supply of material.

Still another possibility for the improvement of the industry in the Silverton district lies in cyanidation. There are silicious gold and silver ores which have proved amenable to this treatment, giving extractions of 90 odd per cent of the gold and 65 per cent to 75 per cent of the silver. An effort is now being made to secure a guaranteed tonnage of these ores to supply a custom cyanide mill and some progress has been made in this direction.

Idaho.

The Lost Packer smelter, which is situated about 100 miles from Mackay, Idaho, the nearest railroad point, was blown in for the summer campaign early in August. This is the first time in three years that this plant has operated, and it is not hard to see that the work is carried on under difficulties. Operations must cease with the advent of winter on account of the impossibility of getting fuel and other supplies. Coke has to be hauled to the smelter and matte has to be hauled out to the railroad, and considering the shortness of the campaign it is rather remarkable that the plant can be operated at all. Since the last operation of the plant a great deal of mining development has been carried on and the prospect is fair for a good campaign this summer.

In the Silver City district of Idaho the Rich Gulch Mining Company is preparing to erect a modern amalgamation and cyanidation plant. For the past six years this company has been opening a large body of low-grade gold and silver ore, averaging about \$11 per ton, and sufficient ore has been blocked out to warrant the erection of the milling plant. The estimated cost for mining and milling is about \$3 per ton, and with reasonably good extraction of the metals there should be a good profit.

Gold, Silver, Copper and Lead in California.

The mine output of gold, silver, copper and lead in California in 1910 had a value of \$27,020,405, according to figures compiled by Charles G. Yale and just made public by the United States Geological Survey. The production of gold was \$19,715,440; that of silver 1,840,085 fine ounces, valued at \$993,646; that of copper 48,700,756 lb., valued at \$6,184,996; and that of lead 2,870,977 lb., valued at \$126,323.

These figures show a decrease in the output of gold, silver and copper as compared with the figures for 1909, but a large increase in lead. The 1909 figures are as follows: Gold, \$20,237,870; silver, 2,098,253 oz., valued at \$1,091,092; copper, 57,288,281 lb., valued at \$7,447,476; lead, 1,502,597 lb., valued at \$64,612.

There were 1079 mines producing gold, silver, copper or lead in California in 1910, of which 564 were gold placer mines. Of the deep mines 485 were gold mines, 9 were silver mines, 10 were silver-lead mines and 11 were copper mines. Of the placer producers 168 were hydraulic mines, 72 were dredges operated by 41 companies, 139 were drift mines in ancient river gravels and 185 were sluicing mines. Measured by the number of producers as well as by tonnage and metal output, deep mining decreased somewhat in California in 1910. Among the placers sluicing decreased also, but dredge and drift mining increased.

In 1910 there were 2,697,885 short tons of ore from deep mines sold or treated in California, with an average total recoverable value of \$6.71 per ton, against 3,041,688 tons in 1909, with an average recoverable value of \$6.41. Of the total recovered value in ores in 1910, milling ores yielded \$10,005,190 in gold and silver; and smelting ores, \$1,669,008 in gold and silver, \$6,184,996 in copper and \$126,323 in lead. Placers produced \$3,888,795 in gold, of which \$635,498 was from hydraulic mines, \$516,929 from drift mines, \$7,550,254 from dredges and \$186,114

from sluicing mines, the yield from dredges alone being over 38 per cent of the total gold output from all sources in California in 1910.

Gold dredging has been of constantly increasing importance in California since 1899, when the yield from this source was \$206,302. In 1910 it was \$7,550,254. The total gold output by dredges in California to the end of 1910 was \$40,318,775. The three great dredging fields are those of Yuba, Sacramento and Butte Counties, of which the Yuba County field showed an increased output of \$730,557 in 1910, while the production from Butte County decreased \$501,038 and that of Sacramento County decreased \$164,542.

Of the total gold placers furnished \$2,468,505 from Butte County, of which \$2,389,235 was from dredges; \$1,394,537 from Sacramento County, of which \$1,369,594 was from dredges; and \$3,199,952 from Yuba County, of which \$3,172,476 was from dredges. Dredges also produced \$191,900 in Calaveras County, \$213,980 in Merced and Stanislaus Counties combined, and \$109,182 in Siskiyou County.

Hydraulic placers produced \$179,105 in gold from Siskiyou County and \$328,154 from Trinity County.

Drift mines produced \$147,599 in placer gold in Nevada County and \$151,743 in Placer County.

Surface or sluice placers produced \$29,700 in Plumas County, \$22,339 in Siskiyou County and \$27,476 in Yuba County, and smaller amounts of gold in most of the other producing counties of the State.

The largest increases in gold production by counties in California were \$347,461 in Amador, \$80,815 in Mono, \$122,363 in Sierra, and \$734,408 in Yuba. The largest decreases were \$499,288 in Butte, \$292,806 in Calaveras, \$78,885 in Mariposa, \$126,752 in Nevada, \$272,940 in Sacramento, and \$310,977 in Tuolumne. The Mother Lode in Amador, Calaveras, Eldorado, Mariposa and Tuolumne Counties produced 1,170,497 short tons of ore in 1910, with a total recovery in gold and silver valued at \$4,435,802, or an average of \$3.78 per ton, against 1,338,160 short tons yielding \$4,854,355, an average of \$3.60 per ton, in 1909.

Arizona and New Mexico.

The porphyry coppers continue to make an excellent record in ore development and milling. It is claimed that the work at the Ray Consolidated is resulting in 70 per cent recovery. Four of the eight sections of the concentrator are now in operation treating about 800 tons each per day, or slightly below their capacity of 1000 tons each. It is probable that no more sections will be placed in commission until after October.

During the past seven months the Inspiration Copper Company has added over 7,000,000 tons of developed ore to its supply. This gives a grand total of over 25,000,000 tons of developed and partly developed ore, averaging 2 per cent copper. Based on a milling capacity of 5000 tons per day, this amounts to about 14 years' supply. Plans for the mill are now being prepared and although a definite site has not been selected the Inspiration will shortly have to be counted among the producers.

The first section of the Chino mill was to have gone into commission before the end of September, treating 1000 tons of ore per day. A good record has been made in bringing this property to the producing state, only two years and three months having elapsed since its acquisition by the present owners. The entire original milling plant of three sections should be in operation before the end of this year and the two additional sections recently decided upon will be ready for operation next year. Over 50,000,000 tons of ore have been developed by means of churn drills. Two years' supply of ore is already stripped and three steam shovels are constantly at work on further stripping operations.

American Mining Congress.

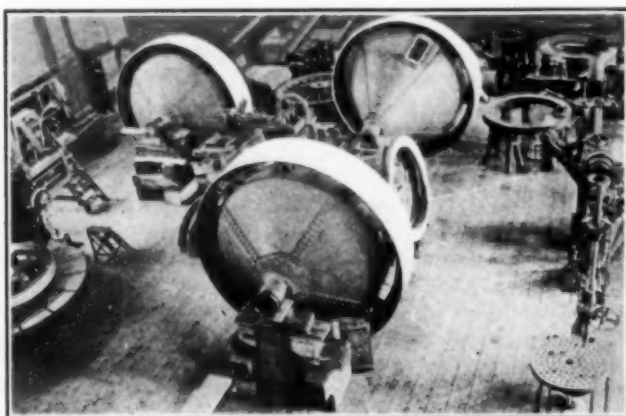
The annual convention of the American Mining Congress will be held in Chicago, October 24 to 28 inclusive. This is

the second change of date that has been made since the first announcement of the meeting, and corresponds with the date originally set. It is expected that President Taft, Secretary Fisher and Director Holmes will be present, and the change in date is for their accommodation. Matters of considerable interest to the metalliferous mining industry will come before this session of the congress, and it is expected that steps will be taken which will result in the national government taking a more direct interest in metallurgical problems, particularly those of the Western states.

Driving of Large Pebble Mills.

A certain innovation has recently been made in the driving of large-diameter pebble mills. The ordinary and common practice has always been gear drive owing to the unusual moment of high power necessary to start a mill from the state of rest.

An innovation was made by the Calumet & Hecla Mining



DRIVING OF LARGE PEBBLE MILLS.

Company, who, after testing out a gear-driven Hardinge 8-ft. conical pebble mill, equipped the 8-ft. mill with an improvised pulley around its largest diameter, extending the pulley diameter to 10 ft. The testing out of this mill under this improvised condition appeared to show that it would take about three to four horsepower in excess of the gear-driven mill. The results of this experiment, however, did not deter the Calumet & Hecla Company from instituting this method of driving this class of mills, which they were then building, as is shown by the enclosed photograph of the Calumet & Hecla shops.

This enormous pulley on a direct-driven mill was obtained with a 26-in. belt. It finally worked out that instead of taking an excess of power there was a decrease in power nearly to the amount as had been expected would be an increase. In other words, while a 35-hp to 36-hp mill under the gear-driven system was expected to require 40 hp, it was found that two of these mills were started from a state of rest with a full load when driven by a 70-hp motor without the motor showing any effect of an excessive load.

It is estimated that these mills are taking in fact from 30 hp to 32 hp.

The point of interest in a mechanical way is the elimination of gears at a sacrifice of a larger belt. It is stated that a belt of about 20 in. in width would give the necessary power. So successful was the operation of the four mills so constructed that the Calumet & Hecla Company are completing the balance of the first installation of about eight to ten more mills under the same conditions.

Arizona Copper Company.

It has been definitely decided to proceed with the erection of the new works which the Board of Directors have had under advisement for some time. Dr. L. D. Ricketts, of the

Cananea Consolidated Copper Company, has been retained as consulting engineer for a period of three years. He will have charge of the preparation of plans and erection of the works, and will be assisted by the company's general manager, Mr. Norman Carmichael. It is expected that plans will be ready to submit to the board this fall.

New Mill for Inspiration.

For some time past there have been reports of a consolidation of the Inspiration and Miami properties, or at least of joint operation of a large milling plant. It appears now, however, that no agreement has been reached and that the Inspiration company will proceed with the construction of its own concentrator. The overtures for joint operation of a mill are said to have emanated from the Miami company, and the suggestion for consolidation from the Inspiration, but the various propositions did not meet with the favor of the two interests.

The capacity of the new Inspiration mill will be 5000 tons per day. The present developed tonnage of ore is 24,000,000 tons and new drill work is adding to this amount monthly.

The Iron and Steel Market.

The steel market has again gone contrary to expectations. July and August proved to be months of fair activity, when extreme dullness had been expected, while September has shown a decline in prices and in activity when improvement was confidently expected. There have been no disorderly breaks in prices, but instead a constant though very gradual sagging, somewhat more rapid in September than in July and August, which was quite natural in view of the reduced demand.

The decrease in activity has been most marked in respect to new buying. Specifications on contracts have been fairly satisfactory, though hardly at as good a rate at the close of September as during August. Buyers are pursuing an extremely conservative course, keeping their stocks down to the lowest possible notch.

A very unfavorable feature of the situation from the standpoint of mill operations is the fact that buyers are ordering and specifying in very small quantities, and the mills themselves have no accumulation of specifications. As a result rolling schedules must be made up almost from day to day, roll changes are frequent and tonnage outputs at the mills are disappointing while the production cost of course is increased. Buyers cannot be induced to place larger specifications, for as it is specifications are obtained only with difficulty, and mills cannot accumulate so as to arrange specifications to advantageous rolling because when material is once specified the buyer insists on the earliest possible delivery.

A rough estimate would indicate that during September the independent steel interests operated at an average of 65 per cent. of their capacity and the United States Steel Corporation at an average of 75 per cent. of its capacity, which means that the domestic business was divided approximately according to the respective capacities, while in addition the steel corporation produced its usual large tonnage of export material.

Finished steel products, with the exception of rails, have receded since July 1 by \$1 to \$2 a ton, plates, bars, sheets and tin plates having declined \$2 a ton and structural shapes, wire and merchant pipe \$1 a ton. These recessions have not been by definite drops but by slight gradations. A trifle more than one-third of the total decline occurred in September. During these three months open market conditions have prevailed, practically for the first time in a decade or more.

The steel industry has lost practically all its optimism. There is little if any expectation of an improvement in demand this year. Conditions are regarded as although unsatisfactory, but the blame is attached more to excess of capacity than insufficiency of demand. The country has been making and consuming pig iron at the rate of about 23,000,000 tons a year, ca-

capacity being slightly in excess of 33,000,000 tons. The best output in 12 consecutive months has been about 30,000,000 tons, from August 1, 1909, to August 1, 1910, but that output was not all consumed. The best previous record was 27,000,000 tons in the 12 months ended October 31, 1907. These are now recognized to have been exceptional periods. The recent rate of output represents the maximum capacity in so recent a year as 1905, when this was the production with all capacity employed and conditions generally satisfactory.

While the continued decline in prices has not brought the average level of finished steel products down to the historic low level, the present average is not more than \$4 a ton above that level, made in 1897 and 1898, at a time when wage and commodity rates were very much lower than at present. The advances in these items amount to much more than \$4 a ton, so that as the steel industry was making very little money in 1897 and 1898 it would be losing heavily at this time were it not that various economies have been introduced. It is recognized that with prices as low as they are now, and showing prospects of further declines, a general reduction in wages can hardly be averted, and reductions averaging 10 to 15 per cent. are likely to be ordered before the close of the year.

FIG IRON.

Southern iron has again become easily obtainable at \$10, Birmingham, a price which almost disappeared in August. Sales are light, consumers having bought more heavily than necessary considering the rate of consumption in the past two months. Foundry iron in other districts is practically unchanged. In the central west deliveries of merchant basic iron have decreased somewhat. A new low price on basic iron at valley furnaces has been made, 1000 tons being sold for prompt delivery at \$13.50, delivered Pittsburgh or \$12.60 f.o.b. valley. Prices are quotable as follows: Birmingham, \$10 for No. 2 foundry; delivered Philadelphia, \$15 for No. 2X foundry, \$14.75 for basic; f.o.b. Chicago district furnaces, \$14.50 for No. 2; f.o.b. valley furnaces, 90c higher delivered Pittsburgh; Bessemer, \$15; basic, \$12.60 to \$13; malleable, \$13; forge, \$13; No. 2 foundry, \$13.50.

STEEL.

The smaller open-hearth steel mills are offering billets and sheets bars at substantial concessions, \$2 a ton below prices prevailing early in August. The large mills making Bessemer steel have important contracts based upon arrangements for monthly adjustment of price, and can not afford to disturb such adjustments by selling at prices the limited amount of steel which could thus be absorbed. The result, due to this peculiar market alignment and in no respect to difference in cost of manufacture or availability for finishing operations, is that open-hearth material is quotable at \$2 less than Bessemer. We quote open-hearth steel f.o.b. Pittsburgh district mills at \$19 for billets and \$20 for sheet bars and Bessemer steel f.o.b. mills Pittsburgh or Youngstown districts, \$21 for billets and \$22 for sheet bars. Rods are nominally quoted at \$27, Pittsburgh, but could probably be bought at \$26.

FINISHED MATERIAL.

Due to open market conditions a considerable range of prices exists on several of the products, according to tonnage and specifications involved in the given order. Prices quoted below are the ordinary minima for desirable orders, but in exceptional cases the prices on bars, plates and shapes are slightly shaded. Small orders and orders for forward delivery generally command slightly higher prices than here quoted. Prices are f. o. b. Pittsburgh, except where otherwise stated:

Rails standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.25 cents for tank quality.

Shapes, 1.30 cents for beams and channels, 3 to 15-in., inclusive, zees, and angles 3 x 3 and larger.

Steel bars, 1.15 cents, base; iron bars, 1.25 cents, Pittsburgh; 1.22 cents, delivered Philadelphia; 1.20 to 1.25 cents, Chicago.

Wire nails, \$1.65, base; plain wire, 1.45 cents, base.

Black sheets, 28-gage, 1.90 cents; galvanized, 2.90 cents; blue annealed, 10-gage, 1.40 cents; painted corrugated roofing, \$1.35 per square; galvanized, \$2.50.

Tin plates, \$3.60 per base box.

The Non-Ferrous Metal Market.

The September market was not characterized by anything unusual either in prices or sales. The foreign situation in zinc continues to indicate a shortage there and an advance in price. The domestic lead market at St. Louis has resumed almost normal relation to the New York market. Spelter and copper have been without feature and prices have been about stationary.

Copper.—There has been a disposition on the part of producers to make concessions in order to get business, and the largest transactions have been at receding prices. The market has been almost stationary at 12½ to 12⅝ cents for Lake, and 12.30 to 12.35 cents for electrolytic.

Lead.—The New York market has held steadily to 4.50 cents since our last report. The St. Louis market has declined somewhat on account of unusually large offerings of Missouri lead, and the latest quotations are 4.35 to 4.40 cents, which is more nearly in accord with the New York price.

Tin.—Prices for spot tin have fluctuated widely in the past month. At one time a premium as high as 4 cents per pound was paid for spot tin in New York, but later this receded to ¾ cents. In general, however, the market has been dull and September tin was quoted at about 41¾ cents.

Spelter.—The stocks of this metal seem to be low and consumers are apparently well supplied. Efforts have been made to create an interest in future business, but the only result has been a tendency toward lower prices. The New York market has stood at about 5.83 to 5.90 cents, and St. Louis at 5.70 to 5.75.

CORRESPONDENCE.

Fixation of Atmospheric Nitrogen.

To the Editor of Metallurgical and Chemical Engineering:

SIR—Referring to my letter concerning the Notodden and Rjukan plants for the production of nitrates from air (published in the September issue, page 436), I have just noticed a mistake in the fourth paragraph. This paragraph should read as follows:

"With reference to the chemical part of the Rjukan plant it had been decided some time ago to use the absorption towers of the Badische Company. It has been decided recently, however, that absorption towers of the Birkeland and Edye type will be used at the Rjukan plant" (instead of Notodden plant, as said in the former letter).

Paris, France.

A. S. GARFIELD.

Aluminium in Ferrotitanium.

To the Editor of Metallurgical and Chemical Engineering:

SIR—In reply to Dr. G. B. Waterhouse's criticisms of the content of aluminium in carbonless ferrotitanium made by the aluminothermic process, I would like to make the following remarks:

The addition of aluminium to carbon-free ferro-titanium, containing 24 per cent of titanium, is intentional and decidedly useful. Titanic acid is well known to be very difficult of fusion, and for this reason the titanic acid which is being formed from the ferrotitanium separates out only with great difficulty, if alone present. If, however, alumina is produced together and simultaneously with titanic acid, the two react together at once, forming aluminium titanate, which is much more easily fusible than titanic acid and also more easily fusible than alumina.

Aluminium, used to such a large extent in the steel industry,

has also the disadvantage that its oxide alone is fusible only with great difficulty and does not separate out easily. By microphotographic methods it is easy to show the small particles of alumina enclosed in steel.

A simple experiment shows that aluminium titanate gives a much more fusible slag than either titanate acid alone or alumina alone. Let us try to melt pure alumina alone and pure titanate acid alone in a piece of clay pot or in a small shallow crucible by means of the oxy-acetylene flame. Either oxide alone will refuse to melt, except if the flame is applied for a very long time and concentrated directly on the oxide, but even then we will succeed only in melting together small globules.

But if we now mix the two oxides together, say in the ratio of three to four parts of titanate acid to one part of alumina (that is, in same proportion in which the two oxides are formed in the oxidation of the carbon-free ferrotitanium, containing aluminium and made by the aluminothermic method), the two oxides together will melt in a very short time. This simple experiment shows clearly the advantages of the aluminium content in titanium.

But there is another important point. Aluminium is a more strongly deoxidizing reagent than titanium. It is wrong to say that titanium is a better deoxidizer than aluminium. This is proven by the fact that aluminium reduces titanate oxide and titanium does not reduce alumina.

The use of the titanium has other objects than the aluminium. While the titanium is a strong deoxidizer, it also reacts with the nitrogen and is effective in small quantity as titanium itself.

When, however, the steel contains quantities of oxides which could be deoxidized by means of aluminium, if this was present, the titanium acts as a deoxidizer, and is consumed uselessly as such wholly or partly. In order to avoid this useless consumption of the titanium aluminium is added simultaneously with titanium, so that all the impurities and oxides are first reduced by the aluminium. When this is done the titanium can manifest its own effect more strongly, and *experience* has shown that to produce the same results *smaller quantities* of ferrotitanium containing aluminium are required than of ferrotitanium containing carbon.

The above-mentioned difficulties with alumina alone in steel have not escaped other experimenters who have tried to avoid and overcome them. I call attention to English patent 6323 of the year 1895, relating to the use of an alloy of manganese, silicon and aluminium, instead of pure aluminium. It is emphasized there that this alloy in statu nascendi yields an easily fusible slag. This alloy is made by the world-renowned firm of Friedr. Krupp in Essen, who are the owners of the above patent.

HANS GOLDSCHMIDT.

Essen-Ruhr, Germany.

Ferrotitanium for Rails and Rolls.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article in your July issue, page 342, by Dr. Hans Goldschmidt is as usual of interest to all who have undertaken to use titanium as a purifier in iron or steel, but I take exception to that part in which the author attributes the success of titanium in rails and rolls to a carbonless alloy.

Statistics compiled by the American Iron and Steel Institute show that 195,940 tons of ferrotitanium rails were rolled during the year 1910. The records of the Titanium Alloy Manufacturing Company show that we shipped a quantity of titanium alloy with carbon to the great steel mills of this country for a considerably greater tonnage than has reached the attention of the Institute.

There are in use in this country to-day very nearly 500,000 tons of titanium rails for which this company has furnished the alloy. Orders on our files indicate that every rail mill in the United States but one purchases the titanium with carbon

alloy made by this company and the one exception is a company which assures us that they have not yet used any ferrotitanium beyond the samples furnished by this company.

Orders for titanium alloy made by us have run as high as 900,000 lb. in a single instance with dozens of orders varying from a carload up to 14 carloads as a maximum, the 900,000-lb. order having required 18 cars of 50,000 lb. capacity.

Under these circumstances it seems evident that a free-from-carbon alloy with its necessarily higher price and its greater tendency to cause brittleness, both due to the very considerable aluminium content cannot possibly have been an important factor in the manufacture of steel rails.

We have proportionately strong evidence on the subject of chilled rolls, but there are more manufacturers, so that of this line we cannot speak quite as positively.

CHARLES V. SLOCUM.

Titanium Alloy Manufacturing Company,
Pittsburgh, Pa.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: With reference to the letter from Mr. Charles V. Slocum on the subject of ferrotitanium for rails and rolls, I would like to state on behalf of Dr. Hans Goldschmidt, who is at present in Germany and therefore not in a position to write a prompt reply on the subject, that I am quite sure that he had no idea of attributing the success of titanium in rails and rolls only to the carbonless alloy.

What he desired to convey was the fact that for this class of work the carbonless alloy possesses very many advantages over a ferrotitanium containing carbon. As he has brought these advantages out in a reply which he has written to Dr. Waterhouse, it is hardly necessary for me to repeat them.

I would like to say further that in Dr. Hans Goldschmidt's article entitled "Melting Point and Its Relation to Alloying Capacity," published in METALLURGICAL AND CHEMICAL ENGINEERING July, 1911, his remarks were entirely general and scientific and related to his personal experience with the carbonless alloy. No one would be more ready than Dr. Hans Goldschmidt to accord Mr. Slocum credit for his energy in introducing his alloy to the steel trade, but his apparent intention to draw the conclusion that the carbonless alloy has found no field of usefulness in the lines he mentions is hardly correct.

WILLIAM C. CUNTZ.

Goldschmidt Thermit Company, New York City.

The Oil Engine for Electrochemical Plants.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your August editorial on "the power problem for electrochemical plants" you have not spoken of the cheapest one of all, namely the oil engine. I have long ago abandoned steam wherever practicable; and the gas engine I have also laid aside after thirty-five years' use, for I find the new F H oil engine built by the De La Vergne Company beats even the gas engine, for with oil at 2.25 cents per gallon, a kw-hour can be obtained for \$0.005 and less, and with less labor and trouble than gas.

How dreadfully slow people are to take up new ideas even when based on scientific principles, to wit: in 1876 when the late Sir W. Siemens first melted steel electrically he said to me, "like everything else it will take a generation before this is brought into practical use"; and this is generally true, for I can point out numerous great ideas which took over thirty years to come into use.

In this western country there are natural conditions which would not lie undeveloped a day in Germany, but I have tried in vain to interest Eastern capital. The answer always comes back: "We know nothing of that kind of business." Oh! for a firm out here like Siemens & Halske. There would be something doing.

WALTER E. KOCH.

El Paso, Tex.

Recent Ore Dressing Practise at Silverton, Col.

Editorial Correspondence.

The rugged mountainous district of southwestern Colorado is generally spoken of as the San Juan region, or simply "the San Juan." It is divided naturally by the mountain ranges into districts which are tributary to the important towns of the region, of which Silverton is one of the most prominent. The winning of gold and silver from the mountains in the immediate vicinity of Silverton dates back to 1860, when gulch mining was first attempted by a party of miners headed by John Baker. But it was not until the early eighties that the deposits of rich silver ore in the vicinity of Red Mountain were discovered and successfully worked. At that time ore worth less than \$100 per ton could not be marketed at a profit on account of the difficulties of transportation. With the extension of the Denver & Rio Grande Railroad into Silverton in 1882 an impetus was given to the mining and smelting of lower grade ores, but with the waning price of silver the precious metal content of the ores still had to be comparatively high in order to yield a profit.

Crude ore only was shipped from this district up to 1890, when pioneer attempts at concentrating the lower grade ores were made at the Silver Lake and Sunnyside mines. Prior to this time lixiviation plants had in a few instances been successful in extracting silver by a process of chloridizing, roasting and leaching with strong brine.

At the present time, however, attention is being directed to the concentration of the low grade milling ores, which are abundant in the district. These are mixed sulphides of lead (galena), zinc (sphalerite), copper (tetrahedrite and chalcopyrite) and iron (pyrite). Silver occurs mainly with the copper ores and sometimes with the finely crystalline galena. Gold accompanies the pyrite and sphalerite in small quantities. Zinc minerals almost always accompany galena, and in the past have given much trouble in concentrating. In brief, operations in the Silverton district are at present confined almost wholly to the separation of these low grade mixed sulphides into marketable products of lead and zinc, and the future of the mining industry there practically depends on the use of the most modern and improved methods of ore dressing.

The presence of considerable quantities of copper and iron minerals with the lead and zinc complicates wet dressing and calls for accessory treatment such as electrostatic separation. The latter process is not now in use in the district and as a consequence many mines and mills are closed, and only those are operating which can produce marketable lead and zinc concentrates by straight water concentration. At present these are the Silver Lake, Vermillion, Ledge, Iowa and Sunnyside. The Vermillion and Ledge mills may be taken as typical of the best ore dressing in the district.

The Vermillion Mill.

The concentrating mill of the Vermillion Mining Company is situated about $1\frac{1}{2}$ miles west of Animas Forks in California Gulch, near the head waters of the West Fork of the Animas River. The crude ore is a mixture of galena, sphalerite and pyrite finely disseminated in a gangue of rhodonite. An average assay will show about 5 to 6 per cent lead, 10 per cent zinc and 3 to 4 per cent iron. The ratio of concentration is about $5\frac{1}{2}$ to 1, and for the month of July, 1911, the saving was 81 per cent. At present the mill is running below capacity and treating only 75 tons per day. Water is obtained partly from the mine, about one-third the total quantity used being taken just as it flows from the tunnel. The balance is pumped from the creek 146 ft. below by a 7x8 Gould triplex pump operated by a 15-hp motor. Pumping is apparently a necessary expense, as the mill is situated so near the head waters of the creek that a pipe line or flume run at the proper level would not collect a sufficient volume of water. From 12 to 13 tons of water are used per ton of ore treated.

The method of treatment follows: Ore is trammed from the tunnel to the head of the mill and dumped into crude ore bins, first passing over a $\frac{3}{8}$ inch grizzly. The coarse ore is reduced to $\frac{3}{8}$ -inch size in a Blake crusher, 11x18 in., and added to the grizzly undersize. The bin capacity is sufficient for the storage of 200 tons of crushed and an equal tonnage of uncrushed ore. From the second bin the ore is drawn by a plunger feeder and belt conveyer to the roughing rolls. These are 14x27 in., running at 60 r.p.m. and set $\frac{1}{2}$ in. apart. The main elevator, which delivers the roll discharge to the trommel screen line, has 55-ft. centers and 14-in. buckets on a 16-in. belt.

A single line of trommels covered with Tyler double crimped screen prepares the feed for jigging. The first screen is 2-mesh, the oversize of which is reground in a set of 14x27 in. rolls running at 125 r.p.m. and set $\frac{1}{8}$ in. apart. This roll product is again elevated to the screens. The second and third screens are 6 and 14-mesh, respectively, and their oversizes are treated in 3-compartment Harz jigs. Owing to the abrasive action of the gangue, screens are rather short lived in this mill. Trommel screens last about three weeks, and it is probable that punched steel screens will be used in place of wire, as local experience indicates that the life of the latter will be about three months. With punched screens the pitch of the trommels will be changed from 1 in. per foot to $\frac{3}{4}$ in.

Jigging practise varies with the physical condition of the ore. At times the mineral does not occur coarse enough to warrant the use of the coarse jig. With both jigs running, however, the first receives a feed ranging from -2 to +6 mesh, and the second from -6 to +14 mesh. The coarse jig has 5-mesh screen in the first compartment, and 6-mesh in the second and third. It runs at 210 r.p.m. and has a stroke of $\frac{3}{8}$ in. The fine jig has 8-mesh screen in all compartments, runs at 235 r.p.m. with a $\frac{3}{8}$ in. stroke.

Both jigs make a clean lead concentrate in the hutch and side gate of the first compartment. A shipping grade of zinc concentrate is made in the third compartment of the fine jig only. The middlings and tailings of the coarse and the tailings of the fine jig are reground in a Chile mill, while the middlings of the fine jig are returned to its first compartment for retreatment.

One 6-ft. Chile mill is used for regrinding jig tailings and middlings. This machine is grinding about 40 tons per day through ton-cap screen corresponding to 14 mesh. The life of the screen is but 10 days due to the excessive abrasion of the rhodonite. The die ring is $4\frac{1}{2}$ in. thick, while the roll tires are made $3\frac{1}{2}$ in. thick on the outside and $3\frac{1}{4}$ in. on the inside. These dimensions are calculated to give about equal life to die and tires. Under the light feed at present given to the mill it is run at only 24 r.p.m.

The Chile mill product is elevated and combined with the undersize of the last, or 14-mesh trommel, and thickened in a 3x3 ft. dewatering box preliminary to classification in a Richards 4-in., 6-compartment pulsator classifier. The overflow of the dewatering box flows successively to two Callow tanks, each of which gives a thickened feed to one Card table. The overflow of the second Callow tank runs to waste.

Each of the six products from the classifier is treated on a separate Wilfley table. The first three tables make high grade lead and zinc concentrates, a two-inch streak of mixed lead and zinc mineral being returned to the tables by raff wheels. The fourth, fifth and sixth tables give a finished lead product, but the zinc mineral from these is combined with the zinc from the Cards treating the Callow product and retreated on another Wilfley. The coarse middling from the first Wilfley is reground in the Chile mill and the middlings from the succeeding five Wilfleys are combined and treated as a mixed feed on a separate table. This last table also has a raff wheel for the return of its middling product. The slime water from all tables except the zinc retreating table is thickened in three Callow tanks and retreated on a Card table.

Table Speeds and Strokes.

	Speed r.p.m.	Stroke in.
No. 1 Wilfley.....	245	$\frac{7}{8}$
Nos. 2 and 3 Wilfley.....	250	$\frac{3}{4}$
Nos. 4, 5 and 6 Wilfley.....	255	$\frac{5}{8}$
Midd. Wilfley	250	$\frac{3}{4}$
Cards	260	9/16

An interesting feature of the table work at this mill is the manner of running the Wilfley tables adopted by Louis D. Bastian, mill superintendent. When the tables were run flat enough to bring all mineral—galena and sphalerite—to the head of the table in the usual manner more or less of the rhodonite gangue appeared scattered throughout the concentrates and the zinc separation especially was poor. Mr. Bastian found that by increasing the pitch of the table so as to bring the zinc mineral over the side, and only the lead over the head of the table, he could increase the grade of zinc concentrate from 37% to 43%, and at the same time throw the rhodonite out of the mineral. Additional wash water is used near the head of the table to assist in washing the rhodonite out of the concentrates, and under present conditions this coarse gangue appears as a narrow streak below the zinc.

Ultimately all concentrates of like character from jigs and tables are mixed for shipment. The average grade of these products is indicated in the following figures:

	Pb %	Zn %	Fe %	SiO ₂ %
Lead Conc.	57-62	7	7	2
Zinc Conc.	8	43	7	6-7

Of the iron in the zinc concentrate a little over half is combined in a manner which precludes separation, but the iron in the lead concentrate represents practically free pyrite. Electrostatic separation, therefore, would be of little benefit to this ore, as the added cost of further cleaning the zinc mineral would hardly be repaid.

The Vermillion mill is operated by electric power supplied by the San Juan Water and Power Company. The distribution of motors in the mill follows: 25 hp for crusher; 75 hp for rolls, trommels and elevator; 30 hp for Chile mill; 20 hp for 10 tables, classifier, two centrifugal pumps and middlings elevator; 15 hp for pump. The total power requirement under running load is 90 hp.

Acknowledgment is made to Mr. L. D. Bastian, mill superintendent, for assistance in collecting the data given and to Mr. R. W. Hollis, manager, for permission to publish the same.

The Ledge Mill.

The ore of the Ledge Consolidated Mining Company at Chattanooga, near Silverton, differs decidedly from that of the Vermillion as to the nature of the gangue, although the mineral content is not much different. Instead of the hard rhodonite gangue noted in the Vermillion ore, that of the Ledge is kaolinite, soft enough to be scratched by the finger nail and making an excessive quantity of slime in the milling process. The minerals are galena, sphalerite and pyrite, the percentage of lead being about 10 and of zinc 12. The ore also carries about 3 oz. silver and 0.02 to 0.03 oz. gold per ton. About 150 tons are treated daily. The ratio of concentration is 5 to 1.

The ore as mined is delivered to the mill by an aerial tramway one-half mile long. The buckets have a capacity of about $\frac{1}{2}$ ton each, and the daily tonnage can generally be delivered on one shift. The ore breaks easily and preliminary crushing is rather coarse to avoid the production of slime. The first break is made to about $1\frac{1}{2}$ in. size in a Blake 10x20 in. crusher. This product is further reduced in rolls, 16x36 in., running at 60 r.p.m. and set $\frac{3}{4}$ in. apart. Water is added at the rolls and the pulp is elevated to the trommel screen line. The elevator has 64 ft. centers and 14 in. buckets on a 16-in. belt. Ample screening capacity is one of the features of the mill,

there being a double line of four trommels, each 9 ft. long and 42 in. in diameter, fitted with ton-cap screens corresponding to 4, 6, 8 and 16 mesh respectively. The oversize of the 4-mesh screen is reground in a set of 14x27 in. rolls, running at 100 r.p.m. and set $\frac{1}{4}$ in. apart and returned to the elevator and screen line. The oversizes of the second, third and fourth trommels comprise the feed for three 3-compartment Harz jigs, and the undersize of the last screen is thickened and classified for table concentration.

The following tabulation gives data regarding the operation of jigs:

	Size of feed Mesh	Screen Mesh	Speed r.p.m.	Stroke in.
Jig No. 1.....	-4 + 6	5	180	$1\frac{1}{4}$
Jig No. 2.....	-6 + 8	8	220	$\frac{7}{8}$
Jig No. 3.....	-8 + 14	10	240	$\frac{5}{8}$

Galena concentrate is drawn from the first compartment in all jigs and from the second compartment also of the third jig. No zinc concentrate is made in the jigs. All jig middlings and tailings are dewatered in a trommel and collected in a bin, from which they are fed to a Chile mill. The water recovered from this pulp is stored in a tank and used in the Chile mill as needed. This procedure is preferable to allowing the pulp to flow from the jigs to the mill without dewatering, as a better ratio of water to pulp can be maintained.

The Chile mill runs at 32 r.p.m. and grinds about 75 tons per day through 16-mesh ton-cap screen. These screens last about three weeks. The die ring is cast thicker than the roll tires, so that all have about the same life, viz., three months.

The performance of the Chile mill at this plant emphasizes the difference in the physical properties of the Ledge and Vermillion ores. The quantity reground at the Ledge through 16-mesh screen is almost twice that reground at the Vermillion through 14-mesh, and yet the screens at the Ledge have about twice the life of those at the Vermillion, where the gangue is rhodonite.

The Chile mill product is elevated and combined with the undersize of the 16-mesh trommel, flowing to a spitzkasten, where it is thickened before classification. On account of the large quantity of slime in the pulp which tends mechanically to carry fine mineral to waste, extra large settling capacity is provided so that all slime is ultimately thickened and concentrated. To this end the overflow of the spitzkasten flows successively to Callow tanks, sufficient in number so that the overflow of the last will be practically clear. The underflow of these Callows is combined and classified in a Richards 4-compartment pulsator classifier and treated on four tables. The thick pulp from the spitzkasten is classified in a Richards 6-compartment pulsator classifier, running under a 16-ft. head of hydraulic water and making 160 pulsations per minute. A hutch product of clean galena is made in this machine, amounting to one ton per day.

The Wilfley tables in this mill are arranged on two floors, ten on each floor. Of those on the upper floor four treat the classified fine material from the Callow tanks and six receive the different grades of coarse pulp. All tables on this floor make a finished lead product only. The zinc-iron middling from each of these tables is retreated on a complementary table on the floor below, where zinc concentrates are separated from the iron and small amount of lead remaining with it. By way of further recovering fine mineral now going to waste it is planned to collect the slime water from the tables on the upper floor, thicken it and reconcentrate on additional tables. It is planned also to regrind the coarser middlings from some of the upper tables. When this plan is in operation the only material going to waste will be the tailings from the lower tables and the overflow from Callow tanks.

All concentrates are laundered to boxes at the foot of the mill and dewatered by a screw conveyer working in an inclined launder at one end of the box. The overflow from the

box passes through a series of settling launders before leaving the mill. The lead concentrates as mixed for shipment assay from 55 to 60 per cent. lead, and the zinc concentrates carry from 40 to 45 per cent. zinc.

During the summer season the mill is operated by water power. Two Pelton wheels running under a head of over 800 feet of water furnish all the power required. One 36-in. wheel, with $\frac{3}{8}$ -in. nozzle, provides power for the concentrating tables only, thus assuring steady operation. The balance of the mill is run by a 56-in. wheel, $\frac{3}{4}$ -in. nozzle. About 150 hp. is required for the whole plant under running load, or one hp. per ton of ore treated per day. Electric power is available when water is scarce, and only a short time is required to make the change.

For the data and information given herewith acknowledgment is made to Mr. W. Z. Kinney, manager, and Mr. D. C. McNaughton, mill superintendent.

Cost of Alkali Chloride Electrolysis.*

BY VICTOR ENGELHARDT.

Aside from the electrolytic production of chlorates, there are two main groups of commercial processes for chloride electrolysis:

(a) *Direct Production of Hypochlorite Solutions.*—For this purpose the products of the electrolysis, chlorine and caustic alkali, are directly combined in the electrolytic cell itself to form bleaching liquor (hypochlorite). There is, therefore, only one final product. Such installations are suitable only in works in which hypochlorite solutions are directly needed.

Since in this case no saturated solutions are employed and the temperature is kept relatively low (as otherwise chlorate would be formed), the power consumption is comparatively high. Further as the ampere-hour efficiency decreases with increasing amount of active chlorine in the hypochlorite solution, it is impractical to decompose the whole chloride so that part of it is wasted.

On the other hand, these installations have the advantage that they need hardly any attendance and that any number of cells can be connected in series so as to suit any given supply voltage. For this reason the cells may be connected to dynamos which are also used for other purposes; for instance, direct-current dynamos which supply the lighting current during the night, may be connected to the bleaching liquor cells during the day.

This last-named practice is especially suitable for the cotton industry and for laundries, where the amount of hypochlorite solution required is relatively small. The electrolytic cells, when operated only during the day, will therefore furnish sufficient chlorine for the requirements of the works, especially as for cotton there is a relatively large saving of active chlorine, if electrolytic hypochlorite solutions are used, as compared with bleaching powder.

On the other hand, in the linen industry and in paper and pulp mills much greater quantities of hypochlorite solution are required, so that in this case it pays in general to install a special dynamo for the electrolytic plant, especially as the latter may then be made of 50 per cent less capacity, being operated continuously day and night. Moreover, in this case there is not such a large saving of active chlorine for hypochlorite solutions as compared with bleaching powder, so that for this reason alone the quantity of hypochlorite solution needed is relatively large.

(b) *Production of Chlorine and Caustic Alkali Separately.*—To keep the chlorine and caustic alkali separate from each other cells of three different types may be used: (1) Diaphragm cells, yielding chlorine gas and a caustic solution containing

some undecomposed chloride, which can be removed completely only with some difficulty during the further treatment; (2) gravity cells (Glockenverfahren) yielding also chlorine gas and generally a caustic solution containing chloride; (3) mercury cathode cells, yielding chlorine gas and a caustic solution free from chloride.

In all processes for the separate production of chlorine and caustic alkali a hot concentrated electrolyte is employed. From the caustic solution produced it is possible in diaphragm and gravity processes to recover the undecomposed chloride during the further treatment; this is unnecessary in mercury-cathode processes. The power consumption is, therefore, lower and the utilization of the chloride is more complete than in the direct electrolytic production of hypochlorite.

For the purposes of this article, which compares the different processes from the standpoint of commercial efficiency, the diaphragm and gravity processes may be grouped together, while the situation may be somewhat different, according to the conditions, for mercury-cathode processes. The latter produce a purer caustic alkali, free from chloride, such as needed for some purposes like the manufacture of metallic sodium or potassium, and which may be sold sometimes at a higher market price. As an offset there are higher capital investment and also mostly higher power consumption, due to the use of higher voltages.

On a large commercial scale processes of all three groups are employed for the production of chlorine and caustic alkali. For industrial works which need chlorine and caustic alkali for their own purposes, diaphragm cells are probably most suitable, since they are simpler and of less delicate construction. Such industrial works may proceed further by either of the following two methods:

(1) If the producer has no use for caustic alkali and no facilities for marketing it, he can combine the chlorine and the caustic outside of the electrolytic cell and produce alkali hypochlorite.* From a commercial standpoint, this method has the advantages of better utilization of the sodium chloride and smaller power consumption (due to both higher ampere-hour efficiency and smaller voltages) over the direct production of hypochlorite in an electrolytic cell.

(2) If the producer can use himself the caustic alkali or if he can market it readily, it is preferable from a commercial standpoint to combine the chlorine gas not with the valuable caustic soda or potash, but with milk of lime (calcium hydroxide) and form a chloride of lime solution.

Installations for the separate production of caustic soda and chlorine in industrial works are suitable only for relatively large capacities, since in order to reduce the relative cost, the apparatus must be built for high current strengths and a sufficient number must be connected in series. Moreover they must be operated continuously day and night, they require their own dynamo, attendance, etc., and would, therefore, be uneconomical for small capacities.

The power cost is of fundamental importance in chloride electrolysis since it is the most important item in the cost sheet. We will assume two limiting values: a minimum value of 0.5 cent per kw-hour for waterpower plants and by-product gas plants, and a value of 1.25 per kw-hour for steam plants.

On the basis of the above discussion the different cases to

*This method is, of course, possible only if what the producer needs in his works, is bleaching liquor. If he needs chlorine gas, this method cannot be used. At the plant of the U. S. R. & R. Co. at Colorado Springs, Colo., where chlorination of gold ores with cyanidation of the tailings is in very successful operation, the chlorine is produced by a diaphragm-cell process and the caustic soda solution is wasted, since there is not such a ready market for caustic soda in the neighborhood as to justify the erection of expensive and troublesome evaporation apparatus. While the waste of the caustic soda seems peculiar from the standpoint of the Eastern producer of caustic soda and chlorine, it is undoubtedly the logical thing to do under the existing conditions. That the chlorine gas can be produced at very much less cost by electrolysis than from bleaching powder, bought in the market, and sulphuric acid, was proven by extensive tests at this plant, using both methods side by side over a considerable period of time.—Editor.

*Translated from *Chemiker Zeitung*, Vol. 35, No. 64 and 65, p. 573 and 582 (1911). Slightly abstracted.

be treated and compared with each other, may be grouped as follows:

I. Direct production of hypochlorite solution at the consumers works.

1A. Smaller installations (cotton industry), operated only during the day. A saving of 30 per cent of active chlorine, as compared with the use of bleaching powder, is assumed.

1B. Larger installations (linen and paper industry), operated continuously during day and night. A saving of 10 per cent of active chlorine, as compared with the use of bleaching powder, is assumed.

II. Separate production of chlorine and caustic alkali.

II. 1. At the consumer's works.

II. 1A. With recombination of chlorine and alkali, to form alkali hypochlorite. Continuous operation day and night.

II. 1B. With production of chloride of lime solution and separate utilization of the caustic alkali. Continuous operation day and night.

II. 2. Separate production of chlorine and caustic alkali on a large scale for selling both products.

II. 2A. Diaphragm processes with production of a caustic alkali containing undecomposed chloride. Continuous operation day and night.

II. 2B. Mercury-cathode process with production of a caustic alkali free from chloride. Continuous operation day and night.

In order to arrive at a fairly satisfactory comparison between the different processes of alkali chloride electrolysis, attention must be paid to the fact that in the separate production of caustic and chlorine these two products are obtained simultaneously from the same raw material so that it is not possible to charge a definite proportion of the cost of operation against the manufacture of either product. On the other hand in hypochlorite manufacture the bleaching liquor solution only is produced.

For this reason, in considering processes for the separate production of chlorine and caustic alkali it will be more correct to deduct the value of the caustic alkali produced from the total cost of operation and to consider the difference as the cost of the bleaching material produced.

Since in industrial plants sodium chloride will probably always be used, it will be sufficient to give the comparative calculations for sodium chloride only. Further it would be wrong to assume always the same values for cost and selling price of sodium chloride, calcium chloride, and caustic soda, since smaller consumers cannot get as favorable prices as large consumers.

I.—Direct Production of Hypochlorite at the Consumer's Works.

In the main the following three electrolytic hypochlorite cells are now in commercial use, differing from each other by the material used for the electrodes: *a*, Siemens & Halske and Kellner, both electrodes being platinum. *b*, Schuckert, one electrode being of carbon, the other of platinum. *c*, Haas and Stahl, both electrodes being of carbon.

The following calculations are based on the system of Siemens & Halske and Dr. Kellner, with which the author is most familiar.

1A: Hypochlorite production at the consumer's works in small installations of the cotton industry, electrolysis being carried out during the day only.

Assumptions: The electrolytic cell is operated with 120 amp. at 110 volts (the power being, therefore, 13.2 kw), for 10 hours a day. 22.2 kilograms of active chlorine daily, 300 days of operation during the year. \$28.75 per 1000 kilograms of chloride of lime at the plant, \$3.75 per 1000 kilograms of sodium chloride at the plant. A saving of 30 per cent active chlorine compared with chloride of lime with 35 per cent Cl.

(Therefore, about 25 kg of active chlorine in a NaOCl solution correspond to about 100 kg of chloride of lime.)

<i>First Cost:</i>	
Platinum electrodes.....	\$1,350
Accessories (cells, pump, etc.).....	1,150
Total	\$2,500

	Cost of kw-hour.	
	0.5 cent.	1.25 cent.
<i>Yearly Cost of Operation:</i>		
13.2x10x300=39,600 kw-hours.....	\$198.00	\$495.00
170x300=51,000 kg Na Cl at \$0.38 per 100 kg.....	191.25	191.25
4% amortisation and maintenance of platinum.....	54.00	54.00
10% amortisation and maintenance of accessories.....	115.00	115.00
5% interest on accessories.....	\$7.50	\$7.50
	\$615.75	\$912.75

Under the assumption of 30% saving of active chlorine, as stated above, the amount of bleaching liquor produced per year is equivalent to 26,700 kg of chloride of lime; hence the cost of the ton of equivalent chloride of lime.....

\$23.05 \$34.18

If for small quantities of chloride of lime the ton is bought at the consumer's works at \$28.75 as assumed above, the production of the bleaching liquor by electrolysis at the works results, therefore, in a saving of \$5.70 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cent, and in a loss of \$5.43, if the kw-hour costs 1.25 cent. Under the above assumptions there will be neither a saving nor a loss if the kw-hour costs about 0.875 cent. In other words, if the power costs less than 0.875 cent per kw-hour, the electrolytic production of bleaching liquor at the works will be less expensive than buying bleaching powder.

It should be emphasized, however, that in this comparison the well-known technical advantages of electrolytic hypochlorite solutions could not be considered, because they cannot be given in figures. Further the present very high market price of platinum must be taken into consideration.

The wages are not considered in calculating the cost of operation since they would represent at least the same amount in bleaching with bleaching powder for the preparation of the bleaching solution, etc.

1B: Hypochlorite production at the consumer's works in larger installations in the paper and pulp industry, with continuous operation day and night.

Assumptions: Electrolytic cells operated at 110 volts, 600 amp (66 kw), for 23 hours a day, 254 kg of active chlorine daily, 300 days of operation during the year, \$26.25 per 1000 kg of chloride of lime at the plant, \$3.75 per 1000 kg of sodium chloride at the plant. A saving of 10 per cent of active chlorine as compared with the use of chloride of lime with 35 per cent Cl. (About 30 kg of active chlorine in NaOCl solution are therefore equivalent to 100 kg of commercial bleaching powder).

<i>First Cost:</i>	
Platinum electrodes.....	\$6,750
Accessories (as in I A).....	5,000
Total	\$11,750

	Cost of kw-hour.	
	0.5 cent.	1.25 cent.
<i>Cost of Operation:</i>		
66x23x300=455,400 kw-hours.....	\$2,277	\$5,692.50
1900x300=570,000 kg sodium chloride at \$3.75 per 1000 kg.....	2,137.50	2,137.50
6% amortisation and maintenance of platinum.....	405	405
10% amortisation and maintenance of accessories.....	500	500
5% interest on accessories.....	250	250
Total	\$5,569.50	\$8,985

Under the assumption of 10 per cent. saving of active chlorine, as stated above, the yearly amount of bleaching liquor produced is equivalent to 255 tons of chloride of lime; hence the cost of the ton of equivalent chloride of lime

\$21.83 \$35.23

Larger installations of this kind can buy the chloride of lime at less cost than assumed before, say at about \$26.25 per ton at the works. Then the production of bleaching liquor at the works by electrolysis results in a saving of \$4.42 per ton of equivalent chloride of lime if the kw-hour costs 0.5 cent, and in a loss of \$8.08 per ton if the kw-hour costs 1.25 cents. Under the above assumptions there will be neither a saving nor a loss if the kw-hour cost 0.75 cent. For power costs be-

low this rate the electrolytic manufacture of bleaching liquor will be less expensive than buying bleaching powder.

With respect to the technical advantages of electrolytic bleaching liquor and wages the same remarks may be made here as were made in IA.

II.—Separate Production of Caustic and Chlorine at the Consumer's Works.

Only diaphragm cell processes are here considered. The cell should be of compact construction and it should be possible to build and operate it economically in relatively small capacities. A cell which has proven successful on a large scale in the chemical manufacturing industry as well as in plants of smaller capacity is the Billiter cell* which has been developed by the Siemens & Halske Company, together with the Kaliwerke Aschersleben. It is probably the latest diaphragm cell which has proven successful in practice. The following figures relate to this cell.

II.—1A: Separate production of chlorine and caustic with subsequent recombination to hypochlorite, with continuous operation in paper and pulp mills.

Assumptions: Electrolytic cells operated with 500 amp at 120 volts (60 kw), 8 kw for other purposes (hence total power 68 kw), 24 hours per day, 337 kg of active chlorine in the hypochlorite (figuring a loss of 10 per cent in absorption, etc.). Eight thousand hours of operation per year, \$26.25 per 1000 kg of chloride of lime at works, \$3.75 per 1000 kg of sodium chloride. A saving of 70 per cent of active chlorine as compared with the use of bleaching powder of 34 to 35 per cent Cl. (About 337 kg of active chlorine in NaOCl solution, are, therefore, equivalent to 1100 kg of chloride of lime).

First cost:

Electrolytic cells	\$7,000.
Conductors and switchboard	525.
Absorption plant and pipe system	1,562.50
Miscellaneous	912.50
Building, foundations	\$10,000.
Concrete tanks	7,500.
Total	\$17,500.

Cost of kw-hour

	0.5 cent.	1.25 cent.
Cost of operation:		
68x8000 = 544,000 kw-hours	\$2,720.	\$6,800.
350,000 kg. sodium chloride	1,312.50	1,312.50
Wages 5 men at \$300	1,500.	1,500.
10% amortization and maintenance on \$10,000 (electrodes, diaphragms, etc.)	1,000.	1,000.
3% amortization of \$7,500	225.	225.
5% interest on \$17,500	875.	875.
Total	\$7,632.50	\$11,712.50

Under the assumption of 10% saving of active chlorine, the hypochlorite produced is equivalent to 366 tons of chloride of lime per year; hence the cost of the ton of equivalent chloride of lime is

\$20.85 \$32.

As in IB, the price at which chloride of lime can be bought may be assumed here as \$26.25 per ton at the works, so that the production of bleaching liquor at the works by electrolysis results in a saving of \$5.40 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cent, and in a loss of \$5.75 per ton, if the kw-hour costs 1.25 cents. There will be neither a saving nor a loss if the kw-hour costs 0.85 cent. For power costs below this rate the electrolytic manufacture at the works will be less expensive than buying bleaching powder.

The use of hypochlorite solutions produced in this way also involves technical advantages. The expense for license for using the patented process is not included in the operating cost in this and in the following cases, since it is not possible to give for this a generally valid figure.

II.—1B: Separate production of chlorine and caustic with subsequent production of chloride of lime solution and use of the caustic soda, with continuous operation in paper and pulp mills.

Assumptions: Electrolytic cells operated with 500 amp at 120 volts (60 kw), 12 kw for other purposes (hence total power

72 kw). Twenty-four hours' operation per day, 337 kg of active chlorine in the bleaching solution (figuring a loss of 10 per cent in absorption, etc.). Eight thousand hours of operation per year, \$26.25 per 1000 kg of chloride of lime at works; \$3.75 per 1000 kg of sodium chloride at works; \$5 per 1000 kg of burned lime, \$4.50 per 1000 kg coal (evaporating seven times its weight of water), if bought in relatively small quantities.

The caustic soda is figured at only \$3.50 per 100 kg NaOH in form of a solution of 35° Be. on account of the higher selling expenses with such comparatively small plants. There is no saving of active chlorine in bleaching. (Hence 337 kg of active chlorine in CaO_2Cl_2 solution are equivalent to 1000 kg of chloride of lime.) Partial recovery of the undecomposed sodium chloride during evaporation.

First cost:

Electrolytic cells	\$7,000.
Conductors and switchboard	525.
Absorption plant and pipe system	1,562.50
Evaporation plant and tanks	2,000.
Miscellaneous	912.50
Buildings, foundations and concrete tanks	\$12,000.
	8,000.

Cost of kw-hour

	0.5 cent.	1.25 cent.
Cost of operation:		
72x800 = 576,000 kw-hours	\$2,880.	\$7,200.
250,000 kg. sodium chloride	937.50	937.50
200,000 kg. lime	1,000.	1,000.
166,000 kg. coal	750.	750.
Wages, 6 men at \$300	1,800.	1,800.
10% amortization and maintenance on \$12,000	1,200.	1,200.
3% amortization on \$8,000	240.	240.
5% interest on \$20,000	1,000.	1,000.
	\$9,807.50	\$14,127.50
Less selling price of 140,700 kg. of caustic soda ..	4,925.	4,925.
Cost of 333 tons of equivalent chloride of lime ..	\$4,882.50	\$9,202.50
Cost of 1 ton of chloride of lime	14.68	27.65

The price at which chloride of lime can be bought may again be taken as \$26.25 per ton, so that the electrolytic operation at the works in the present case results in a saving of \$11.57 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cents, and in a loss \$1.40, if the kw-hour costs 1.25 cents. There will be neither a saving nor a loss if the kw-hour costs 1.175 cents. For power costs below this rate the electrolytic manufacture at the works will be less expensive than buying bleaching powder.

Compared with the former examples, the power may cost considerably more if electrolytic manufacture at the works is to be profitable. This is due to the utilization or sale of the caustic soda.

II.—2. Large-scale separate production of chlorine and caustic soda for sale of the products.

Since in this case we have to do with large plants, relatively low prices may be assumed for the purchase of the raw materials. On the other hand a somewhat smaller selling price must be assumed for chloride of lime. The following figures relate to the Billiter cell for diaphragm-cell processes, while platinum anodes are assumed for the mercury-cathode processes, since figures from experience with the mercury-cathode process with carbon anodes are not yet available.

II.—2A: Diaphragm process with chloride of lime and solid caustic soda (containing some sodium chloride) as final products with continuous operation in large-scale plants of the chemical industry.

Assumptions: Power consumption: 1200 kw for electrolysis, 30 kw for chloride of lime apparatus, 30 kw for evaporating plant, 20 kw for the pumps, ventilators, etc., 70 kw for lighting and miscellaneous, hence total power consumption 1350 kw. Voltage 3.75, amp-hour efficiency 90 per cent, 24 hours of operation during the day, 8000 hours per year. Selling price of chloride of lime in large quantities \$20 per ton. Selling price of solid caustic soda (95 per cent), \$47.50 per ton, including packing. Purchase price of sodium chloride \$3, lime \$4.50, coal \$4.50 per ton.

*See this journal, Vol. VIII, p. 704 (1910).

First cost:

Electrolytic cells.....	\$125,000	
Copper conductors.....	13,500	
Pipes.....	13,500	
Chloride of lime apparatus.....	62,500	
Evaporating plant.....	20,000	
Melting pots.....	5,000	
Screening and lime burning plant.....	3,000	
Plant for purification of water, solution of salt and lye transport.....	5,750	
Cranes and other transportation facilities.....	1,250	
Packing material factory.....	4,000	
Workshop, motors and gears.....	7,750	
Sum A.....		\$261,250
Lighting and heating plant.....	\$4,500	
Equipment of office and laboratory.....	4,000	
Sum B.....		8,500
Buildings, C.....		87,500
Building erection, superintendence and miscellaneous, D.....		30,250
Working capital, E.....		37,500
Total.....		\$425,000
(Without cost of land and railroad track connections.)		

Cost of operation:	Cost of kw-hour	
	0.5 cent.	1.25 cent.
(a) Power: 1350x800 = 10,800,000 kw-hours.....	\$54,000	\$135,000
(b) Raw materials: 5505 tons of salt.....	\$16,515	
6086 tons of lime.....	27,387	
6000 tons of coal.....	27,000	
	\$70,902	71,000
(c) Renewal and maintenance: Electrodes.....	\$14,000	
Diaphragms.....	2,000	
Electrolytic cells.....	1,250	
Chloride of lime plant.....	4,375	
Evaporating plant.....	2,000	
Melting pots.....	5,000	
Buildings.....	1,750	
Balance of equipment.....	4,375	
	\$34,750	35,000
(d) Wages: 58 men at \$375.....	\$21,750	
3 foremen at \$750.....	2,250	
	\$24,000	25,000
(e) Auxiliary materials: Oil, lubrication, belts.....	6,000	6,000
(f) Packing, excl., wages: Caustic soda.....	\$7,000	
Chloride of lime.....	20,000	
	\$27,000	27,000
(g) General: Wages.....	\$6,250	
Office and laboratory.....	3,000	
Insurance.....	2,500	
	\$11,750	11,750
(h) Capital amortization: Average, 7%, since all amortization has already been taken into account.....	28,500	28,500
(i) Interest: 5% on \$425,000.....	21,250	21,250
Total (not including patent license).....	\$279,500	\$360,500
The yearly production of caustic soda is 3,560 tons (95%).....	\$169,100	\$169,100
Balance for the yearly production of 8,690 tons of chloride of lime.....	\$110,400	\$191,400
Or per ton of chloride of lime.....	12.70	22.33

If, as assumed, the selling price of the ton of chloride of lime is \$20, there is a profit of \$7.30 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour, and a loss of \$2.33 per ton for a power cost of 1.25 cents per kw-hour, assuming an interest of 5 per cent only on the capital. For a power cost of 1.075 cents per kw-hour there will be neither profit nor loss, that is, just the 5 per cent of interest on the capital will be realized.

II.—2B: Mercury-cathode process, with chloride of lime and solid caustic soda (free from sodium chloride) as final products with continuous operation in large-scale plants of the chemical industry.

Assumptions: Power consumption 1600 kw for electrolysis, 150 kw for other purposes, together 1,750 kw. Voltage (at least) 5 volts. Ampere-hour efficiency, 90 per cent; continuous operation for 24 hours per day and 8000 hours per year. Selling price of chloride of lime in large quantities \$20 per ton. Selling price of solid 95 per cent caustic is to be assumed higher than in II 2A, namely \$50 per ton, including packing. Purchase price of sodium chloride \$3, lime \$4.50, coal \$4.50 per ton.

First Cost:

Electrolytic cells without platinum and mercury.....	\$62,500	
Copper conductors.....	13,500	
Pipes.....	13,500	
Chloride of lime plant.....	62,500	
Chloride of lime plant.....	20,000	
Melting pots.....	5,000	
Screening and lime burning plant.....	3,000	
Plant for purification of water, saturation, and lye transportation.....	9,000	
Cranes and other transportation facilities.....	1,250	
Packing material factory.....	4,000	
Workshop, motors, and gears.....	8,750	
Sum A.....		\$203,000
Lighting and heating plant.....	\$4,500	
Office and laboratory.....	4,000	
Sum B.....		8,500
Metals (mercury and platinum) C.....		137,500
Buildings D.....		87,500
Building erection superintendence and miscellaneous E.....		26,000
Working capital F.....		37,500
Total.....		\$500,000
(Without cost of land and railroad track connections.)		

Cost of operation:	Cost of kw-hour.	
	0.5 cent.	1.25 cent.
(a) Power: 1,750 x 8,000 = 14,000,000 kw-hours.....	\$70,000	\$175,000
(b) Raw materials: 5,505 tons of salt.....	\$16,515	
6,086 tons of lime.....	27,387	
5,300 tons of coal.....	23,850	
	\$67,752	\$67,750
(c) Renewal and maintenance: Metals.....	\$5,500	
Electrolytic cells.....	1,250	
Chloride of lime plant.....	4,375	
Evaporating plant.....	2,000	
Melting pots.....	5,000	
Buildings.....	1,750	
Balance of equipment.....	6,150	
	\$26,025	\$26,000
(d) Wages (as in II. 2 A).....	25,000	25,000
(e) Auxiliary materials (as in II. 2 A).....	6,000	6,000
(f) Packing (as in II. 2 A).....	27,000	27,000
(g) General (as in II. 2 A).....	11,750	11,750
(h) Capital amortization in the average 7% with the exception of platinum and mercury which keep their value, hence on about \$375,000.....	26,250	26,250
(i) Interest of 5% on \$500,000.....	25,000	25,000
Total (not including patent license).....	\$284,750	\$389,750
Sales price of 3,560 tons of 95% caustic soda at \$50 per ton.....	178,000	178,000
Balance for yearly production of 8,690 tons of chloride of lime.....	\$106,750	\$211,750
Or per ton of chloride of lime.....	\$12.28	\$24.38

With a selling price of \$20 per ton of chloride of lime and \$50 per ton of caustic soda and an earning of only 5 per cent of interest on the capital investment, there is a profit of \$7.72 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour and a loss of \$4.38 per ton for a power cost of 1.075 cents per kw-hour. For a power cost of 0.975 cent there will be neither profit nor loss, that is, just the 5 per cent interest on the capital investment will be realized.

II.—2. B': If, however, with several large plants of this kind the caustic soda cannot be sold at \$50 per ton, but only at \$47.50 (as in II 2A), then the results are changed as follows:

Cost of operation, yearly.....	Cost of kw-hour.	
	0.5 cent.	1.25 cent.
Selling price of 3,560 tons of 95% caustic soda at \$47.50 per ton.....	\$169,100	\$169,100
Balance for yearly production of 8,690 tons of chloride of lime.....	\$115,650	\$220,650
Or per ton of chloride of lime.....	\$13.33	\$25.40

With a selling price of \$20 per ton of chloride of lime and an earning of 5 per cent interest on the capital investment, there is a profit of \$6.67 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour, and a loss of \$5.40 for a power cost of 1.075 cents per kw-hour. For a power cost of 0.925 cent per kw-hour there will be neither profit nor loss, that is, just the 5 per cent interest on the capital investment will be realized. It appears, therefore, that the mercury-cathode processes can complete with diaphragm processes only at very low power cost, say about 0.25 cent per kw-hour.

The results of the preceding seven cost calculations are summarized in form of the adjoining diagram. This diagram gives the cost of the production of the ton of chloride of lime

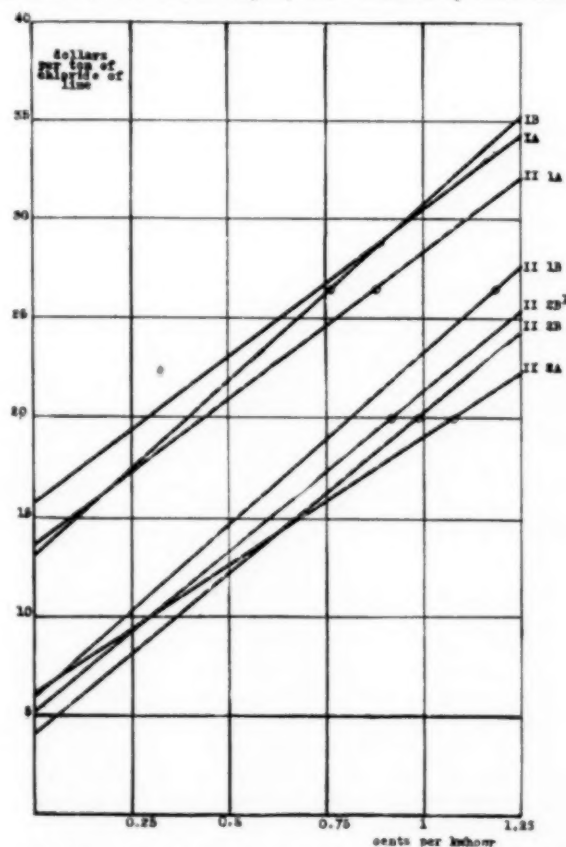
(or in cases IA, IB, and II 1A, of the equivalent ton of chloride of lime) as function of the power cost.

The ordinates give the cost of the ton of chloride of lime in dollars, the abscissas the cost of the kw-hour in cents.

The curves have been plotted by using the following figures which give the end results of the preceding calculations.

	0.5 cent.	1.25 cents per kw-hour.
I A	\$23.05	\$34.18 per ton of chloride of lime.
I B	21.83	35.23 "
II 1 A	20.85	32.00 "
II 1 B	14.68	27.65 "
II 2 A	12.70	22.33 "
II 2 B	12.28	24.38 "
II 2 B	13.33	25.40 "

The points in the different curves for which under the assumptions made in the different cases we break even financially with alkali chloride electrolysis, are indicated by little circles.



COST OF ALKALI CHLORIDE ELECTROLYSIS PROCESSES AS FUNCTION OF POWER COST.

The parts of the curves at the left of these circles, therefore, represent the conditions under which alkali chloride electrolysis is financially profitable.

If it is desired to use other values than those assumed for the cost of the raw materials or the selling price of caustic soda, the above calculations can be easily changed accordingly.

As the market conditions are at present [in Germany] for the raw materials and final products, we can sum up our results in the following conclusions:

1. According to the different processes, the maximum power cost for which alkali chloride electrolysis is profitable, varies between 0.75 and 1.125 cent per kw-hour.

2. In the smaller installations of the textile industry the direct electrolytic production of hypochlorite (electric bleaching) is in general financially profitable only if a certain saving of active chlorine is figured on. But even if the cost of producing the bleaching liquor by electrolysis is somewhat higher than with the use of bleaching powder, the other technical advantages of electrolytic hypochlorite (less wear and tear of the fibres, smaller acid consumption, etc.) still justify the use of the electrolytic hypochlorite process.

3. In paper and pulp mills as well as in larger installations of the textile (especially linen) industry, it will in general be more profitable not to produce directly a hypochlorite solution, but to produce separately chlorine and caustic soda and make use of or sell the caustic soda. With satisfactory power prices, the erection of such plants is justified *per se*, as against the purchase of chloride of lime.

4. In the large-scale chemical industry and under average conditions the diaphragm processes are superior to the mercury-cathode processes with respect to both first cost and cost of operation. In the case of erection of a new plant, the use of a mercury-cathode cell will be justified only if there is a sure market for the caustic soda, free from sodium chloride, at a higher selling price, and if very cheap power is available.

My thanks are due to Mr. Ebert and Dr. Carl for their help in collecting the data and in checking the numerical calculations.

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Gold Milling in California.

BY WILLIAM H. STORMS.

At the time of the discovery of gold in California the methods and means of milling gold ores were of the crudest possible description. In fact, aside from the operations of Mexicans, with arrastras, and South Americans, with their Chile mills, about all that was known of the science of recovering gold from its ores by wet processes was the extremely inefficient methods that had been in vogue in the region about Dahlonega, Georgia and at a few other places in the South, where mining had been attempted in a most primitive way.

Whatever extremely limited ideas the early California miners may have applied to the conduct of their business were patterned after either the practice of the Southerners or that of the Mexicans or Chilenos. Among the first mills built and operated in California in 1850-51 was one at, or near, what is now Nashville, in El Dorado County. This affair consisted of a battery of five stamps. The stems were square and were made of wood. These were lifted by nearly straight cams which operated in vertical slots cut in the stems, the involute cam having not yet been discovered by the mill men. The lower ends of the stems were fitted with iron-shod square shoes. The mortar consisted of a plate of iron laid in the bottom of a wooden box, iron-bound and secured with bolts. This was placed on a foundation of timbers laid at right angles, forming a solid crib. The power was supplied by a small steam engine.

No copper plates were used, the gold being caught in riffles and on heavy blankets, which latter were washed frequently. It is almost needless to say that this crude affair was a most unsatisfactory gold-saving contrivance, nor that for the greater part of the time it was hung up for repairs.

About the time that this mill began operations a party of prospectors discovered a rich vein outcropping a few miles up the canyon from the mine at Nashville. The owners of the mill above described had a duplicate of it at their place but had not set it up, and when the discoverers of the new mine heard of this they at once opened negotiations for the purchase of the outfit, which it appears the owners were glad enough to sell. The mill was moved up to the new discovery and there set up, but not without sundry improvements over that down the canyon.

The purchasers of this mill, who were the owners of what has since been known as the Union mine, took no pains to deny that they were proud of their new acquisition. This mill, although set up with all the skill, care and knowledge that their limited experience suggested, proved to be but little, if any, better than that down the gulch at Nashville.

Soon after starting up all sorts of unsatisfactory conditions

developed. A set of square iron shoes were made for it at Sacramento, but were found to be no better than the others. One of the men suggested an iron stem and shoe, which idea was favored by the other members of the company, but the departure was too radical from the established practice to take the risk of fitting up the entire battery in this new fashion, so they compromised by arranging for one stamp to be placed in the center between the wooden ones.

This iron stamp was made from a piece of shafting and was so evidently superior to the wooden affairs that it was promptly decided to replace all of the wooden stamps with those of iron, but no suitable bars of round iron could be obtained in either Sacramento or San Francisco at the time, and it was several months later before the necessary iron arrived around the Horn at San Francisco.

When the new battery of iron stamps, with round bosses, shoes and dies was finished and set to work, it proved to be such a marked improvement over any of the cumbersome wooden affairs that all mill builders promptly adopted the innovation. In this new mill the wear and tear of cams and tappets was excessive, as the stamp was prevented from turning by a device arranged for the purpose.

It was then suggested by one of the owners, a Mr. Sanford, I think, that it would be an improvement if the stamp were allowed to turn as it was raised by the cam. Some thought not, but the experiment was tried and of course worked to perfection. The cam "dope" was pine tar, which was smeared on with a stick and quickly spread evenly over the under face of the tappet.

Thus one detail after another in the primitive mill was improved as the necessity for improvement appeared. Each detail was worked out by experiment, there being no other school in which to learn than that of hard and expensive experience. An innovation in any mill that proved a success was promptly copied in every mill along the lode, until, in the course of a few years, the California stamp-mill, as we know it to-day, was evolved by these indefatigable pioneers in the business.

After a time copper plates were tried and proved a great success, but it was not until after 1860 that silver-plating of the copper plates was introduced, the first of these, I think, being on the Comstock Lode. Self-feeders were first attempted about 1858, and although they were a great improvement over hand feeding, it was many years before they reached anything like their present perfection.

The mechanism of the stamp-mill has undergone a great change since those early days—far more than the mill practice in California, at any rate, for, with few exceptions, the mill arrangement and daily routine is much the same to-day as it was 30 or more years ago. One would naturally suppose that after all these years of experimentation and varied practice we would have tried out now about every device and that we should have long since been able to plan the ideal gold mill, a mill that all would be willing to agree was superior to any other, yet such is not the case.

The ideal mill is rarely seen, for the ideas of mill builders—not so much the mill founders as those who buy mills and have them built after their own plans and specifications—are almost as widely different as they ever were.

Even to this day can be seen mills where the rock-breaker sets above the level of the floor, so that every pound of ore that goes through the machine must be lifted on a shovel by the attendant. Still more rarely, fortunately, may be seen the spectacle of a man feeding the battery with a shovel. It is but fair to say of this, however, that it nearly always is due to lack of means to purchase an automatic feeder, rather than to the old-fashioned notion that the work could be done better by hand.

It is generally acknowledged that the rock-breaker should be placed outside the mill building and on a separate foundation, but if the situation be such that the breaker must be placed at

the mill it is considered good practice to at least provide it with an independent foundation and separate framework of timbers entirely detached from the mill structure. This I have seen done at no small expense, and the independent timber frame then solidly connected with the mill structure by means of timbers and bolts, thus undoing all the advantage that might have been derived from the separate breaker-foundation arrangement.

Ore bins have come in for their share of discussion, some contending that the bin with a flat floor is the best, as it admits the storage of a greater or less amount of ore and also tends to give steadiness to the mill structure. Others maintain that the bin with the sloping floor is best because it supplies the feeders with ore by gravity, still others take the middle course and favor a bin having a bottom which is flat at the front and sloping at the back.

It is my opinion that the proper place to store broken ore is either in the mine or in a bin outside the mill and that the ore bin in the mill should have a sloping bottom from the front backward at an angle of at least 45 deg. to the back of the bin, from which level it may very properly be vertical. In a bin of this shape all the ore delivered to it will run by gravity to the chute doors, excepting, possibly, a very small amount that may lodge in the angle at the bottom between the chutes.

Flat bins permit the storage of ore, it is true, but when this stored ore has to be drawn upon, whenever the natural slope-angle down which the ore will run by gravity has been reached, men must be put into the bin to pick and shovel the ore to the chute doors, and this can rarely be done under an average cost of 20 cents per ton, and it is generally more. Flat bins may have some advantage for the purpose of storage where the ore is high grade, but where the matter of 20 cents per ton is a serious thing for the manager of a low-grade mine to consider, as it is at most low-grade mines, the flat ore bin has no place in arrangements looking toward economy.

On the other hand, few managers of high-grade mines care to keep a lot of rich ore tied up any longer than necessary, so there seems to be no legitimate place for flat ore bins. I can see no advantage of the bin built with a narrow flat section longitudinally of the bin just back of the chute doors. It looks like a compromise between two extremes with nothing whatever to recommend it.

Mortars vary greatly in shape. Some are wide, some are narrow and there is considerable difference in the slope-angle of the interior of mortars near the bottom. All of these variations have a greater or less effect on the rate of discharge, and also to a great extent they determine the condition in which the pulp is discharged from the mortar. A narrow mortar will discharge more rapidly than a wide one, and so, also, will a flaring one. The wide, deep mortar, with high discharge, will slime successfully if this is what is desired, but if a more granular pulp is wanted then the narrow mortar of what is commonly known as the Homestake pattern is the thing.

All ores do not amalgamate with equal facility, and when the type of mortar to be placed in the new mill is to be decided upon this important matter should receive due consideration. If the ore amalgamates readily and most of the gold present is free, then a mortar that will discharge rapidly is necessary and amalgamation may be carried on both inside the mortar and on the outside plates. In some mills large capacity is the main object and no inside amalgamation is attempted. With some exceptionally free-milling ores this is entirely satisfactory, but with other ores is less so.

Ores which are to be amalgamated and the sulphides concentrated after amalgamation should be crushed only sufficiently fine to free the particles of gold and the sulphides from the gangue, further crushing results in the creation of an undesirable quantity of slimes, which will interfere with both amalgamation and subsequent concentration.

All of these peculiarities of ores should be thoroughly in-

vestigated before the type of mortar is decided upon, but in many cases the old-time mill man argues: "A mortar is a mortar, and so long as it is heavy enough to stand the blows of the falling stamps, what more can you ask?" I should say very much more, as I have endeavored to show above. I once had a millman say to me, after an argument over his method of working, "I have been here thirty-five years, and I guess I know what I am doing." This was an unanswerable argument, so I could only venture that he had been at it long enough, and that it was time for him to retire and give some other poor fellow a chance.

Among the numerous other subjects that have been up for controversy is that of battery foundations, and even now, after sixty years of wrangling, and no end of practical experience at home and abroad, not only the old-time millman, but even the engineers, are warmly discussing the various kinds of battery foundations, their shape and size. I think it may be safely said that, except perhaps with the ultra-conservative, concrete foundations are admitted to be superior to those of any form of wood construction.

But at the present moment it is the manner of placing the anchor bolts in these foundations that is being threshed out. I think anchor bolts should be of ample size, say, at least $1\frac{1}{2}$ in. diameter, and supplied with heavy threads that will not strip, these to be so placed in the foundation as to be always easily accessible for tightening, and so arranged as to direction that a broken or worn bolt may be withdrawn at any time and replaced by a new one. To accomplish this its direction in the foundation must be such that it is not obstructed by any portion of the mortar nor by the woodwork of the battery frame. If this be done there appears to be little left to dispute about, as far as I am able to see.

The arrangement of battery posts and their braces is sometimes influenced by the topographical situation, which in some measure may determine the position of the engine, or other motive power. What is known as the "back-knee" construction is the most generally in use, and is the most satisfactory, under usual conditions, but occasionally it is more convenient to place the line shaft on a level with the cam-shaft floor and in front of the line of battery. Here the "front-knee" frame is employed. It should be avoided wherever possible, in favor of the back-knee frame, which is more stable and gives greater rigidity to the battery frame throughout. The front-knee frame is seldom seen in new mills, its undesirable features having been recognized by mill builders generally.

The manner of distributing power from the engine, water-wheel or motor varies, but is of less importance than some of the foregoing considerations. Line shafts are found below the mortars, both behind and in front of the batteries, and at various elevations within certain limitations. Occasionally it is seen on the cam floor, as previously stated, in front of the battery. The best arrangement I have seen was where the line shaft was back of the battery foundations and below the mortars, being supported on a line of independent concrete piers, at a height of about 20 in. above the floor.

Where concrete foundations are to be used I can see no objection to providing a buttress-like projection at the back of these foundations, and at a proper height, to carry the line shaft of the mill. This will render the shaft always easily accessible and will also permit the desirable down-pull on the cam shaft pulley, and should prove an altogether satisfactory arrangement.

I know of but one mill in California where the line shaft has been placed higher than the cam shafts, and this is in the North Star mill, near Grass Valley. There the shaft has been placed at the top of the front side of the concrete ore bin. I fail to see that this presents any advantage over that of placing the shaft below the battery at the back, if, indeed, the North Star idea is as good as the latter.

There is little material difference in the arrangement of the amalgamated copper plates in front of the batteries these

days. Formerly plates divided by a central strip, thus making two sluices each about 22 in. wide, were much in vogue, but these have given place to wider plates, which are far more satisfactory. Even now in many mills the plate area is too limited for the best economic results.

It may safely be said that as long as gold can be collected at the foot of a run of plates it is good business to supply additional plate surface. Where tailings are as low as 25 or 30 cents per ton, additional plate area may reduce the loss sufficiently to justify the expense of their installation, and the extra work of caring for them, but no amount of additional plates will be of service unless they be kept in proper condition.

Every square foot of plate in the mill should be kept as bright and clean as industrious rubbing will make them. Some millmen have a stated time for doing all millwork. This is a mistake. The time to dress plates is when they need it, no matter whether it be twelve hours after the last dressing or in half an hour after. This will make extra work for the millman, but that is what he is there for.

Any portion of a mill plate covered with sand, or that is discolored by mineral stain of any sort, is not catching a particle of gold, and the mill may as well be without that plate surface thus affected.

A 5-stamp battery, crushing 5 tons daily per stamp, will in a year crush 9000 tons of ore. If an additional saving of 5 cents per ton can be effected by the provision of more plate area, it should be done, as it would mean a net saving of \$450 per year for each 5-stamp battery.

In nearly all California gold mines the gold occurs only partly free, the balance of it being associated with the sulphides of the base metals, iron and copper pyrite, and to a less extent with arsenopyrite, galena and blende, and still more rarely with tellurium. This usually necessitates the concentration of the auriferous sulphides.

Occasionally mines are found where the amount of sulphides present is either too small, or of too low a grade to justify their concentration. At the Yellow Aster Mine, in Kern County, for instance, the quantity of sulphides is both too small and of too low grade to warrant their being concentrated; the Sheep Ranch Mine, which at one time was also a large producer, is another example where the sulphides were too poor to concentrate, although the ore averaged \$15 per ton in gold.

The ore of the Golden Gate Mine, near Sonora, in Tuolumne County, on the other hand, carried nearly all the value in the sulphides, amalgamation in the mill yielding only about 50 cents per ton on an ore worth from \$5 to \$20 per ton. The sulphides from this ore were concentrated on vanners and canvas tables and the gold extracted by the Plattner chlorination process.

It is in the process of concentration after amalgamation that the California millman is, generally speaking, behind the times. There are a few exceptions, but they are lamentably few, indeed. However, there appears to be a disposition to investigate other methods, with a view to improvement. This may be, and probably is, due to the entry of younger men into the profession. These younger men are not yet in the rut, and their schooling is very likely to keep them out of it. As a class they are ambitious, energetic, and seeking methods to bring about needed reforms in an antiquated practice, which all who are unbiased will admit are needed.

Ordinarily, from the mill plates the pulp flows by launder to mercury traps, and from these again by launder to the concentrating machines, which are of many makes, but mostly either vanners or tables of the Wilfley type. Some mills have both. In nearly every mill the practice is to split the stream of pulp from the trap below each 5 stamps, into two equal streams, each of these two streams going to a separate vanner or table. No classification of any kind is attempted, and the result is that there is invariable loss.

This loss may be low, it is true, perhaps not more than 30 to 60 cents per ton, but on ore as easily treated as the most of that on the Mother Lode of California this loss is too high, and it can, in probably every case, be lowered if the pulp be classified before going to the concentrating machines.

Two or even three classes of material should be made by the classifying cones, spitzkasten, or whatever device is employed, and each class sent to a separate belt machine, or table, that is carefully adjusted as to vibration, grade, speed of travel, and volume of pulp to water, so as to save the highest possible amount from each class of material under treatment.

This is a physical impossibility in the present practice, where no classification whatever is even attempted. The additional expense would be in the first cost and in maintenance, but the additional saving, in most cases, and I believe in every case, would amply justify the cost.

The average California mill has not been planned with this object in view, and as a result there is generally too little head room between the plate floor and the concentrator floor of the mill. The difference in height rarely reaches 15 ft. and very often it is less than 10 ft., and these heights do not permit the placing of satisfactory classifying devices of any sort in their proper place.

This, however, does not make proper classification in such a mill an impossibility, for there are numerous mechanical devices for cheaply elevating mill pulp, and where there is insufficient head room the difficulty can be overcome by employing one of these in connection with a system of suitable launders to distribute the classified material from the classifiers to the concentrating machines. In Colorado this very important part of gold milling is much better understood than in California, and in the former State may be seen many mills which would serve as an excellent copy for the mills of California.

There is one innovation in mill construction to which I wish to call attention, and this is found in the mill of the Original Amador Gold Mining Company, at Amador City. In this mill, which was built in 1909, there are twenty stamps in two units of ten stamps each. The separate units consist of five stamps arranged on either side of the cam-shaft pulley, this requiring four battery posts to each ten stamps, and a cam shaft a little longer than usual, but the even distribution of stress more than makes up for the strangeness of the new departure, and I think that as soon as the advantages of this style in construction are appreciated there will be many more mills built along similar lines. In this mill also are found classifying devices—one of the very few places in California where they can be found.

Berkeley, Cal.

The process of making sand-lime brick was invented in Germany, where it probably has had its greatest development up to the present.

Observations made there seem to prove that sand-lime brick is a building material which will resist shining soot. Three years ago an inn was built in Greifenhagen, in which sand-lime brick was used for everything except the lining of the chimney flues, which were built of common brick because at that time the question whether sand-lime brick was equally resistant to fire had not been settled. In spite of the fact that lignite briquets were burned, shining soot was found in the flues in a short time, and the flue bricks were permeated as high as the rafters of the ground floor. When the flues were pulled down, however, it was found that the sand-lime brick were unharmed and white in color.

This experience seems to indicate that sand-lime brick will be found in general practice the best building material to withstand shining soot.

The Determination of Gold and Silver in Copper.

BY EDWARD F. KERN AND ALBERT A. HEIMROD.

The determination of gold and silver in copper bullion is accomplished by either the direct-fire assay method or else by a combination wet and fire method. There seems to be no conventional standard method, as at some plants the former is used whereas at other plants the latter one is used, or else a more or less modification of either.

The authors, having undertaken an investigation which required a standard method for the determination of gold and silver in copper-gold and copper-silver alloys, encountered the difficulty of finding a method which would give duplicate check results. The reason for the lack of agreement in the assays, which were made by these methods, could not be charged against anything definite, so the working out of an accurate method was undertaken.

As the chances for losses of gold and silver by volatilization are greater by the direct fire method, and because the proper temperature conditions of scorification are difficult to control, the combined wet and fire method was favored for the determination of the gold, and an entire wet method for that of the silver.

Assay of Copper-Gold Solutions.

One of the main sources of error in the determination of gold in copper bullion (or an alloy), by the combined wet and fire method is the loss of some of the finer particles of gold when the bulky copper solutions are filtered. A means of preventing this loss is to add to the solution some salt that will produce a heavy precipitate and which while precipitating will collect all of the suspended particles of gold. The salts generally used for this purpose are either lead acetate which added to the solution containing sulphates gives a precipitate of lead sulphate, or else silver nitrate which added to the solution containing a chloride gives silver chloride. Whether the one collecting agent possesses any advantages over the other was not known, so the first series of assays was conducted with the idea of finding out which salt is the more suitable one to use.

With this point in view, a solution of gold and copper was made up, each cubic centimeter of which contained 0.17654 gram of copper and 0.000205 gram of gold. Samples of 50 c.c. each of this standard solution were taken and assayed. The general method of procedure, after the sample had been measured out, was to add 10 c.c. of concentrated sulphuric acid and evaporate to dryness. The mass was then taken up with distilled water and diluted to 250 c.c. Ten grams of ferrous sulphate crystals ($\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$) dissolved in distilled water was added, which precipitated the gold in solution as finely divided metallic particles. In order to collect these particles, the first reagent used was from 10 c.c. to 20 c.c. of a saturated solution of lead acetate which gave a heavy precipitate of lead sulphate. The solution was filtered and the precipitate washed, dried, and scorified with 20 to 40 grams of test lead. The button was then cupelled and the gold bead weighed. Results of these assays are given in Table I.

It was found, however, that more satisfactory results were obtained when a silver nitrate solution was used instead of lead acetate, and the silver precipitated as a chloride. The beads obtained by scorification and cupellation, when silver nitrate was used, were parted, and the gold annealed before weighing.

The table (No. 1) shows the amount of collecting reagent used and the results obtained. Although the results on the whole are fair, the errors are not constant; the greater variations occurred when lead acetate was used, and the lesser variations when silver chloride was the collecting reagent. The results, however, gave assurance that for the assay of copper-gold alloys, the latter collecting agent would give the most ac-

TABLE I. ASSAYS OF STANDARD COPPER-GOLD SOLUTIONS.

Assay Number	c.c. of Lead Acetate Solution	c.c. of Conc. HCl Added	Grams of AgNO ₃ Added in Solution	Grams of Test Lead for Scoring	Theoretical Amount of Gold in Solution Gram.	Weight of Gold in Gram Obtained	Difference in Weights Gram.	Percentage Error
1	20	20	0.01026	0.00816	0.00210	20.4
2	30	20	0.01026	0.00982	0.00044	4.28
3	10	25	0.01026	0.01026	0.00000	0.0
4	10	25	0.01026	0.01004	0.00022	2.14
5	10	25	0.01026	0.01024	0.00002	0.19
6	...	5	0.5	25	0.01026	0.01010	0.00016	1.50
7	...	5	0.5	25	0.01026	0.01026	0.00000	0.0
8	...	5	10.0	40	0.01026	0.01017	0.00009	0.87
9	...	5	10.0	40	0.01026	0.01015	0.00011	1.08
10	...	5	10.0	40	0.01026	0.01015	0.00011	1.08
11	10	40	0.01026	0.01013	0.00013	1.26
12	10	40	0.01026	0.01000	0.00026	2.53
13	10	40	0.00718	0.007	0.00018	2.50
14	10	40	0.01026	0.01020	0.00006	.59
15	10	40	0.01026	0.01016	0.00010	0.97

curate results. With this as a starting point, the working out of a standard method for the determination of gold in copper was undertaken.

Preparation of Standard Copper-Gold Alloys.

The standard alloys were prepared by melting from 1500 to 2000 grams of electrolytic copper in a graphite crucible, the copper being completely covered with charcoal. A calculated amount of gold was added to five to ten times its weight of molten copper in a small graphite crucible. After maintaining the alloy in a molten condition for a few minutes, it was poured into the larger crucible containing the major portion of the copper. The larger bulk of copper was first thoroughly poled to tough pitch and then the rich alloy added. The alloy was well stirred with a stick of graphite, and then poured from an elevation of from 6 to 8 feet into a tub of cold water, having a depth of 12 inches. It was found that by granulating the alloy, segregation of the gold (and of silver) was overcome. These granules, varying from less than 1 mm. to 1 cm in diameter, were the standard alloys used in the experiments.

Assay of Copper-Gold Alloys.

An alloy (No. 1), containing approximately 100 oz. of gold to the ton of copper was used, for the first series of assays. As close to an assay ton (29.166 grams) as possible was weighed out, and dissolved in 90 c.c. of concentrated nitric acid. The residue of gold particles were then dissolved by adding 2 c.c. of a mixture of 4 parts nitric acid and 1 part hydrochloric acid (aqua-regia). When all the metal was in solution, 37.0 c.c. of concentrated sulphuric acid were added and the solution carefully evaporated to dryness. The mass of crystals was dissolved in distilled water, and the solution diluted to 300 c.c., then 10 grams of ferrous sulphate crystals ($\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$) dissolved in water was added in order to precipitate the gold.

In the third assay, of Table II, the fine particles of metallic gold were collected in filterable form by adding 10 c.c. of a saturated solution of lead acetate and 0.5 grams of silver nitrate dissolved in water. This gave a mixed precipitate of lead sulphate and silver chloride. In the other experiments the gold particles were collected by adding 0.5 grams of silver nitrate dissolved in water and precipitating the silver with a solution of hydrochloric acid. In all cases the solutions were filtered, the precipitate washed, dried, and together with the filter paper were scorified with 45 grams of test lead. The resulting buttons were cupelled and the beads parted with nitric acid. Results are tabulated in Table II.

The errors in this series of assays (No. 1 to 7) were attributed to the fact that the ferrous sulphate did not throw down all the gold, as it was partially oxidized by the presence of aqua-regia. The reason for adding aqua regia was in order to get the alloy entirely in solution, thus finally obtaining a complete separation of the gold from the copper. By follow-

TABLE II. ASSAY OF STANDARD COPPER-GOLD ALLOYS.

Assay Number	Alloy Number	Weight of Sample in Grams.	Grams of $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$ Added	c.c. of HCl Added	Grams of AgNO_3 Added	Grams of Test Lead for Scoring	Weight of Gold in Gram.	Oz. Gold per Ton
1	1	29.1667	10	5	0.5	45	0.09577	95.77
2	1	29.1709	10	5	0.5	45	0.09482	94.80
3	1	29.1696	10	5	0.5	45	0.09758	97.57
4	1	29.1660	10	5	0.5	45	0.09622	96.22
5	1	29.1660	10	5	0.5	45	0.08932	89.32
6	1	29.1712	10	5	0.5	45	0.09780	97.77
7	1	29.1685	10	5	0.5	45	0.09725	97.24

ing this procedure it was thought that if no copper was present with the gold during cupellation, the errors of assaying would be reduced.

In the next series of assays (No. 8 to No. 13), the gold was not dissolved in aqua-regia, and only silver nitrate used as the collecting agent for the gold particles, it being precipitated as chloride with hydrochloric acid. The errors in this series of assays were accounted for by the presence of aqua-regia formed by free hydrochloric acid and traces of nitric acid in the solution. Table III shows the tabulated results. Alloy No. 1 contained approximately 100. oz. of gold per ton, and alloy No. 2 approximately 10. oz. of gold per ton.

TABLE III. ASSAY OF STANDARD COPPER-GOLD ALLOYS.

Assay No.	Alloy No.	Weight of Sample in Grams.	c.c. Conc. HNO_3 for Dissolving	c.c. Conc. H_2SO_4 for Evaporation	Grams of $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$ Added	c.c. Conc. HCl Added	Gram of AgNO_3 Added	Grams of Test Lead for Scoring	Gram of Gold Obtained	Oz. Gold per Ton
8	1	29.1711	90	37	10	5	0.5	45	0.09912	99.10
9	1	29.1708	90	37	10	5	0.5	45	0.09929	99.27
10	1	29.1680	90	37	10	5	0.5	45	0.08936	89.35
11	2	29.1697	90	37	10	5	0.05	45	0.00990	9.90
12	2	29.1723	90	37	10	5	0.05	45	0.01008	10.07
13	2	29.1722	90	37	10	5	0.05	45	0.01030	10.29

The next set of experiments gave us the method which was adopted for the determination of gold in copper-gold alloys.

Standard Method for Determining Gold in Copper-Gold Alloys.

The method finally adopted and used for assaying copper-gold alloys is as follows: About 1 assay ton (29.166 grams) of the alloy was carefully weighed, placed in a 900 c.c. beaker, and 20 c.c. of distilled water added. The beaker was covered with a watch glass and 90 to 100 c.c. of concentrated nitric acid (sp. gr. 1.42) was added in small quantities of 20 to 25 c.c. from time to time. After all the alloy was dissolved, 40 c.c. of concentrated sulphuric acid (sp. gr. 1.84) was added in order to transform all the copper nitrate into copper sulphate. (0.87 c.c. conc. H_2SO_4 theoretically required for 1 gram of copper). Light blue needle-like crystals were formed by the addition of the sulphuric acid, but were easily broken up by stirring and by subsequent evaporation of the supernatant liquid to sulphur trioxide fumes.

The solution was evaporated to dryness and fumes of sulphur trioxide by placing the beaker on a sand bath, so regulating the heat as to prevent bumping. This left a mass of copper sulphate crystals absolutely dry with part of the water of crystallization driven off. The beakers and contents were allowed to cool, and 500 c.c. of distilled water was added, which readily dissolved the salts on heating. Then 10.0 grams of ferrous sulphate crystals ($\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$), dissolved in water, was added for the purpose of destroying any free nitric acid present and to precipitate any gold in solution. Twenty-five c.c. of a saturated solution of sodium chloride, and enough silver nitrate to make a bead weighing about eight times the weight of gold present were next added. If a greater amount of silver nitrate is used, the bead obtained by scorifying and

cupelling will be too large to part, that is, the gold will be left as a powder instead of a single porous mass.

1.5 gram AgNO_3 is equivalent to 1.0 gram Ag.

The silver nitrate was added when the copper sulphate solution was hot, and the solution rapidly stirred in order to make the silver chloride coagulate and settle, thus collecting all the finely divided gold in the precipitate on the bottom of the beaker. The solution was filtered and the precipitate washed until all copper salts were removed from the filter paper. The beaker was then thoroughly wiped with a piece of filter paper which was added to the main precipitate.

The silver chloride precipitate containing the gold, together with the filter paper, and about 20 grams of test lead were placed on a litharge glazed scorifier. The filter paper was slowly burnt off, and about 20 grams more of test lead added, making a total of 40 grams. A pinch of borax glass was placed on top of the lead and the scorification made in the usual manner. The resulting lead button was cupelled on a medium hard bone-ash cupel, and the gold-silver bead parted with dilute nitric acid (1 part conc. HNO_3 and four parts water). The residue was washed with distilled water, annealed and weighed.

Glazed scorifiers were used in order to prevent loss of gold by particles of the ignited filter paper becoming attached to the sides of the scorifier which prevented their collecting in the lead button. The melting of the glaze washed the particles of ash into the lead bath, which collected any gold which it contained. The glazing was done by putting from 10 to 15 grams of litharge on the scorifier, placing same in a muffle furnace and heating to fusion of the litharge; the excess of litharge was then poured off and the scorifier allowed to cool for subsequent use.

Results of assays made by the above method are given in Table IV. Alloy No. 2 contained approximately 10 ounces gold per ton, alloy No. 3 approximately 36 ounces per ton, and alloy No. 4 approximately 96 ounces per ton.

TABLE IV. ASSAY OF STANDARD COPPER—GOLD ALLOYS.

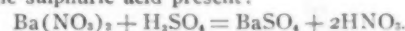
Assay No.	Alloy No.	Weight of Sample in Grams.	c.c. Conc. HNO_3 for Dissolving	c.c. Conc. H_2SO_4 for Evaporation	Grams of $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$ Added	Gram of AgNO_3 Added	Grams of Test Lead for Scorification	Gram of Gold Obtained	Oz. Gold per Ton
14	2	29.1652	90	40	10	0.08	40	0.01044	10.44
15	2	29.1594	90	40	10	0.08	40	0.01040	10.40
16	2	29.1676	90	40	10	0.08	40	0.01042	10.42
17	2	29.1630	90	40	10	0.08	40	0.01040	10.40
18	3	29.1534	90	40	10	0.00	40	0.03568	35.69
19	3	29.1511	90	40	10	0.30	40	0.03560	35.62
20	3	29.1538	90	40	10	0.30	40	0.03566	35.67
21	4	29.1601	90	40	10	0.80	40	0.09583	95.87
22	4	29.1598	90	40	10	0.80	40	0.09583	95.87
23	4	29.1627	90	40	10	0.80	40	0.09584	95.85

Determination of Silver in Copper-Silver Alloys.

The standard copper-silver alloys were prepared in the same manner as the standard copper-gold alloys, as outlined previously.

Samples of the alloy weighing approximately five, ten and twenty grams were used for the assays. The amount of alloy taken in each case depending upon the quantity of silver the alloy contained. The samples were dissolved in concentrated nitric acid (3 c.c. per gram of alloy) in a covered beaker, the excess of acid was evaporated off, and the evaporation continued until copper nitrate commenced to crystallize out of the hot solution. At this point all of the oxides of nitrogen, which are harmful to the precipitation of the silver by means of alkali sulphocyanate, were removed. The solution and salts were taken up with 400 c.c. of cold distilled water, and 25 c.c. of a saturated solution of ammonium sulphocyanate (25 grams per 1 liter) was added to the cold solution as quickly as possible and with constant stirring. The silver in the form of silver sulphocyanate coagulated and settled rapidly to the

bottom of the beaker. The solution was filtered, and the precipitate washed free from copper salts by means of distilled water. The filter and precipitate were placed in the original beaker in which the precipitation was made and digested with 80 c.c. of concentrated nitric acid. The beaker was covered with a watch-glass and placed on a hot plate and allowed to remain until all of the paper was decomposed and the precipitate was entirely dissolved. The solution was evaporated to small volume before all of the oxides of nitrogen were driven off, which is essential for the success of the analysis (did not take to dryness). After cooling, the salts were taken up with 300 to 350 c.c. of distilled water, and 20 c.c. of a saturated solution of barium nitrate was added, which precipitated the sulphuric acid present:



The presence of sulphuric acid is the result of oxidation of silver sulphocyanate by nitric acid.

About 3 cu. cm. of a saturated solution of ferric alum was added, as an indicator, and the solution titrated at ordinary temperature with a standard solution of ammonium sulphocyanate (3.2 grams NH_4CNS per liter). The standardization was obtained on samples of pure electrolytic silver, dissolved in nitric acid, boiled, and the solution diluted to about 100 c.c.

1.0 c.c. NH_4CNS solution was equivalent to 0.004465 gram silver.



The end point is readily distinguished by the appearance of a faint orange color, which the solution assumes as soon as all the silver is precipitated, due to the formation of ferric sulphocyanate.

The accuracy of this method, as described, was first tested on standard copper-silver solutions as follows, and later on on copper-silver alloys. About 60 grams of the standard alloy was dissolved in 180 c.c. of concentrated nitric acid in a covered beaker, and the oxides of nitrogen expelled by evaporation. The solution was diluted to 1000 c.c. with distilled water and three samples of 300 c.c. each were accurately measured out and assayed as already explained. These solutions were titrated with two standardized ammonium sulphocyanate solutions, namely: A solution 1.0 c.c. of which was equivalent to 0.004465 gram of silver was run in to within about 1 c.c. of the end point, then the titration was finished with a second standard solution, 1.0 c.c. of which was equivalent to 0.000584 gram of silver.

It was thought that by using the two standard solutions the determinations could be made more accurately. This procedure was, however, found to be unnecessary, so in the later titrations only the stronger standard solution was used (1.0 c.c. equivalent to 0.004465 gram silver). The results of this series of determinations are recorded in Table V.

TABLE V. ASSAY OF STANDARD COPPER—SILVER SOLUTIONS.

Assay Number	Grams of Standard Alloy in 1 Liter Solution	c.c. of Solution Titrated	NH_4CNS SOLUTION		Gram of Silver in 300 c.c. Solution
			1.0 c.c. = 0.004465 Gram Ag.	1.0 c.c. = 0.000584 Gram Ag.	
1	60.0083	300	18.9	12.0	0.09140
2	60.0083	300	18.9	12.3	0.09157
3	60.0083	300	18.9	12.1	0.09146
4	60.0091	300	19.0	10.8	0.09120
5	60.0091	300	19.0	10.7	0.09108
6	60.0091	300	19.0	10.8	0.09120

The results of determinations made on the copper-silver alloys are given in Table VI. The method of analysis was carried out as described.

In order to obtain accurate results by this method of assaying copper-silver alloys, the following precautions should be observed:

If all the oxides of nitrogen are not expelled, and if the solution is too hot during the precipitation of the silver, then the solution will not retain a green color after the excess of

TABLE VI. ASSAY OF COPPER-SILVER ALLOYS.

Assay Number	Alloy Number	Grams of Sample Taken	NH ₄ CNS Solution. 1.0 c.c. = 0.004465 Gram Ag.	Gram of Silver in Sample	Oz. of Silver per Ton
7	6	20.0279	17.50	0.078140	113.76
8	6	20.0187	17.45	0.077914	113.51
9	7	10.0023	40.40	0.180386	525.99
10	7	10.0036	40.45	0.180609	526.57
11	8	5.0306	50.60	0.225930	1309.90
12	8	5.0213	50.60	0.225930	1311.50

ammonium sulphocyanate has been added, when the silver is precipitated from the copper solution. In the first case the solution will change to the original color, namely, blue, at the same time brown fumes of nitrogen tetroxide (N_2O_4) will be given off, causing some of the silver to go back into solution. In the second case on adding the sulphocyanate to the hot solution, copper is reduced and a precipitate of cuprous sulphocyanate forms which interferes with the accurate titration of the final silver solution.

If the final solution, obtained by dissolving the silver sulphocyanate and the filter paper in conc. nitric acid, is evaporated too far (during the operation of expelling the nitrogen oxide fumes) a brown coating of burnt organic matter from the filter paper forms on the bottom of the beaker, and when the solution is diluted it assumes a brownish color which interferes with detecting the end point of the titration.

The dissolving of the alloy, the evaporation of the resulting solution, the dissolving of the precipitate, and the final evaporation of this solution should be carried out carefully in a covered breaker, in order to prevent error as result of spattering.

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Exploitation of Scandinavian Water Powers.—The *London Electrical Review* of April 1 reports from Berlin that negotiations are being conducted by an Anglo-German-Scandinavian financial group with regard to the consolidation into a company of a number of Norwegian and Swedish water-powers, under the title of the Hydraulic Power and Smelting Corporation, and with a share capital of \$5,000,000 and debentures of \$5,000,000. The scheme relates to water falls already in use and yet to be used, as well as to some of the establishments supplied by them. The assets to be brought into the undertaking include the zinc smelting works and zinc refinery at the Trollhattan Falls and in Sarpsborg, and the water-power of 126,000 hp at the mouth of the Tysse mountain stream in the Hardanger Fiord. The latter installation is already equipped or in process of being equipped, for 57,000 hp and the turbo-generators have been used for years past for supplying the carbide and cyanamide works in the neighboring Odda, in which the German Siemens group is interested. As concerned with the scheme in Germany are mentioned the Allgem. Elek. Ges. the Siemens-Schuckert Works, and the Berlin Electrical Enterprises Co., together with the banks associated with them under the lead of the firm of S. Bleichröder.

Nevada Consolidated Copper Company.—In the quarterly report of this company covering the second quarter of 1911 it is shown that an unusually low grade of ore was handled owing to the advisability of milling a large tonnage of ore removed for the purpose of arranging permanent tracks into Eureka pit. The grade of ore milled during the quarter averaged 1.59 per cent copper. During June all operating reverberatories were fired with oil, except one which was fired with coal and oil. The results have fully justified the use of oil instead of coal at the Steptoe plant. The first basic lined converter was scheduled to go into commission about Aug. 1.

Notes on the Chemistry and Metallurgy of Vanadium—III.

BY WARREN F. BLEECKER.

The Metallurgy of Vanadium Ores.

In the United States the metallurgist may be called upon to treat the following vanadium bearing ores:

Carnotite.
Impregnated Sandstone.
Patronite.
Roscoelite.
Vanadinite.

In addition to the above there have been described a number of vanadium minerals which have not as yet been found in commercial quantities. These are:

Brackebuschite, a lead-zinc vanadate; Dechnite, a lead vanadate $Pb (ZnCu) V_2O_6$; Descloizite, a lead-zinc vanadate with formula $4 (PbZn) O \cdot V_2O_5 \cdot H_2O$; Endlichite, a lead vanadate with $V : As :: 1 : 1$; Eusynchite $(PbZn)_2V_2O_8$, massive descloizite; Tritochlorite, a lead-copper vanadate; Mottramite, a copper-lead vanadate with formula $(CuPb)_2V_2O_8 \cdot 2H_2O$; Psittacinite, a pulverulent copper-lead vanadate; Pucherite, a vanadate of bismuth with formula given as $BiVO_4$; Ramirite, a lead-copper vanadate, and Volborthite, a copper-calcium vanadate with formula given as $(CuCa)_2V_2O_8$.

The pure carnotite mineral is said to contain 18 per cent V_2O_5 and 56 per cent U_3O_8 . It occurs as impregnated sandstone, the ore being usually yellow or yellowish green. If the vanadium content of the ore is low the color is canary yellow, while if the vanadium content predominates (which often occurs) the color may be a dark green, almost black.

Carnotite ores are by far the most soluble of vanadium bearing ores under metallurgical treatment. Decomposition may be effected by roasting with sodium chloride or fixed alkalis, after which a large percentage of each metal is soluble in water. The ore is quite soluble in boiling alkali or alkaling carbonates, there being a patented process based upon this property. Decomposition of the ore also may be effected with mineral acids, sulphuric being preferably used. A very comprehensive report of a proposed process for the treatment of carnotite with sulphur monochloride has been published. The difficulties of treatment are therefore not in decomposition, but in the final separation of the uranium and vanadium values and the economical production of compounds or alloys suitable to the consumer. Although it is not the purpose of this paper to discuss the metallurgy of uranium, it may be said that the behavior of the two elements with reagents is in many respects so different that very little study should at least result in an efficient method of separation; and with the uranium once freed from vanadium there will be little difficulty in getting it into a desirable form.

Sandstone impregnated with vanadium has become an important source of vanadium. The ore is somewhat complex and a dried specimen disintegrates with ease into three general components. The first is a green fine substance which has been carefully studied by Hillebrand; second, a coarse white quartz product, apparently containing some vanadium, and third, a further cementing material consisting of vanadium compounds containing lime, silica, alumina and fixed alkalis, together with probable traces of other elements.

In metallurgical processes the usual procedure is to grind to 20 or 30 mesh and roast with sodium chloride. The best results are apparently obtained by roasting at a temperature of 1100-1200° Fahr. Depending upon the oxidizing conditions in the roast, the product, when cold, may be colored yellow, red, green or even black. If the temperature is allowed to run too high the mass will fuse together, with the result that vanadium compounds insoluble in water are formed. The roasted ore is leached with water by a counter-current system of enriching weak liquor by passing it over fresh ore, and finally

washing the ore with fresh water. The solution cannot be enriched in this particular manner beyond a certain limit, about 8g per liter, because of reprecipitation brought about by the soluble basic matter in the ore. I have obtained the best results by keeping the solutions alkaline. One of the methods I use consists in adding a small quantity of crude lye to the sodium chloride before mixing with the raw ore. This has the disadvantage, however, of decomposing a considerable amount of silica, which must be accounted for later on.

Sodium carbonate may be used for decomposition instead of sodium chloride, but does not give good results unless the ore is of especially high vanadium content.

Merely washing the roasted ore with water will not result in complete extraction in any event. The reasons for this are apparent if it is remembered that the ore also contains a small amount of iron, quantities of lime, alumina and silica. Iron vanadate, calcium vanadate and aluminium vanadate are all insoluble in water, while sodium vanadate will unite with soluble silica to form a compound insoluble in neutral solutions. I arrived at the latter conclusion by mixing solutions of sodium silicate and sodium vanadate and neutralizing with hydrochloric acid. A white precipitate results which, while somewhat gelatinous, may be washed quite readily. Repeated washing did not remove the vanadium.

A good commercial extraction of the vanadium values is obtained if the water leaching is followed by washing with either dilute hydrochloric or sulphuric acid. This has the disadvantage of extracting large quantities of lime and alumina.

It may be said that the fine green substance mentioned above as one of the three components of the ore is much more difficult to decompose than the other two, although the fine is originally the richest in vanadium.

I have read several statements relative to the roaster gases containing chlorine and vanadium. A number of careful examinations of the gases from a muffle furnace under different conditions of oxidation failed to show the presence of either free chlorine or vanadium in any form. Free hydrochloric acid, ferric chloride and sodium chloride were the principal compounds evolved during the roasting when sodium chloride alone was used.

In the treatment of vanadiferous sandstone the chief difficulties are overcome if a good extraction has been obtained, and this may only be done by long and careful experimenting with the particular grade of ore to be treated, because the varying amounts of impurities in the different grades of ore will modify the extraction to a greater extent than might at first be imagined.

Patronite is a carbonaceous sulphide of vanadium occurring with iron, nickel, silica and small amounts of other impurities. There is said to be a large body of this ore in Peru, and it is now being economically treated in the United States by the American Vanadium Company. I have recently received a small sample of non-carbonaceous vanadium sulphide from San Miguel County, Colorado. Analysis showed the presence of other vanadium mineral in addition to the sulphide, since the vanadium content was 15.98 per cent V_2O_5 , while the sulphur content was 4.30 per cent.

The sulphide ore, after roasting to the oxide, may be decomposed with mineral acid, preferably sulphuric. Only a fair extraction is probably obtained, but since the ore is of such high grade the present methods may be the most economical in the long run. Extraction also may be made by roasting the oxidized ore with crude lye and leaching with water. The problem in this case will be to keep the solutions strongly alkaline without using an excessive amount of alkali, although this may be done by any one of a number of regenerative methods.

Where the metallurgist is having difficulty with extraction, I think it is a good rule to use alkalis, since the behavior of the alkaline vanadates is much more uniform than that of the vanadyl compounds, and in most cases the alkaline vanadates

are much more soluble in water, while in dilute solutions vanadyl compounds may be more readily precipitated by impurities than the alkaline vanadates.

Roscoelite is an impossible sort of a mineral to work with commercially. The only agent which I have found to give good results in effecting decomposition is sodium peroxide. Commercially this is out of the question. The best extraction with allowable reagents I obtained by using a mixture consisting of 50 per cent each of sodium chloride and crude lye. Three hundred pounds of this mixture per ton of ore resulted in an extraction of 60 per cent of the vanadium. The raw ore assayed about 3.50 per cent V_2O_5 . The mineral is but slowly attacked by concentrated mineral acids, aqua regia not effecting a complete decomposition upon several hours' boiling. Among the reagents tried with roasting were sodium chloride, sodium hydroxide, nitre cake and sodium carbonate. If there were not much greater quantities of more easily treated ores, or if the available quantities of roscoelite were greater a method of treatment would surely be developed, but at the present time it hardly seems worth while. An additional difficulty in the treatment of roscoelite is presented in the saving of the noble metal content, which in Boulder County ore is often of greater value than that of the vanadium.

Vanadinite is a more or less important mineral, the formula being given as $3Pb_3 \cdot V_2O_5 \cdot PbCl_2$. This is the only vanadium ore (at least of present commercial importance) which can be treated by concentration. A company proposing to operate at Cutter, N. M., claims to have considerable quantities of vanadinite, the concentrates containing 15 per cent $BaSO_4$. I am indebted to Mr. Herman A. Prosser for this information.

Lead vanadate may be decomposed by concentrated sulphuric acid with the formation of divanadyl sulphate under the commercial conditions of treatment. It is apparent that the extraction will not be complete and that there will be large quantities of impurities in the tank liquors. The difficulties of treating ore with sulphuric acid in large quantities render such processes unattractive, although it is claimed by those who are more familiar with the sulphuric acid methods of treatment than I that no difficulties are met with and such processes are satisfactory. In order finally to get rid of the greater part of the impurities, the concentrated vanadyl sulphate solutions are evaporated and finally calcined with sodium carbonate with the formation of sodium vanadate. A good extraction may be obtained by treating vanadinite with strong, hot solutions of lye. The ore must be ground to extreme fineness and the leaching continued for some time. Lead and other impurities are more or less soluble in the mill solution, but the matter of purification is very simple as compared with the purification of acid liquors.

Sodium chloride does not seem to give good results as a decomposing agent for vanadinite, although a fair extraction may be obtained by chloridizing in a muffle furnace.

Numerous patents have been taken out both in this country and abroad for the extraction of vanadium from the various ores and slags in which vanadium occurs. I have never read of a process that seemed more than moderately efficient under the conditions described by the author. Since vanadium has many laboratory peculiarities that are not apparent on a mill scale, most of the processes devised are almost totally inoperative. For example, one inventor has tried to patent a method for recovering vanadium from acid solutions by boiling. The phenomenon of vanadic acid precipitating from a boiling acid solution is of course familiar to all chemists, but those who have taken the trouble to compare the amount of vanadium in solution to that precipitated have not wasted money on trying to patent a process based on the reaction. Again, there have been patented processes for the extraction of vanadium by boiling the ore with sulphuric acid. As a metallurgical operation this has never appealed to me on account of the mechanical difficulties; easy enough in a test tube, perhaps, with five times as much acid as ore, but less easy in tanks of

standard materials and with the human equation of the workmen forever to be taken into account.

Smelting methods have been patented, but since vanadium is essentially a slag-forming element there does not seem to be much hope in that direction, especially in the treatment of ores carrying as low as 1 per cent V_2O_5 . Although there have been patented processes for the extraction of vanadium by fusion with various agents, such as nitre, potassium bisulphate and the like, the tremendous excess of reagent required, together with the additional heat to effect fusion, and finally the difficulties of dissolving the fused material, should make one very cautious in recommending such a method.

Chemical Methods Necessary.

After the entire field of theory, backed up by thorough experimental evidence, has been canvassed, the metallurgist must be convinced that the sole method of attacking the problem of vanadium extraction is chemical. Expensive reagents cannot be used, nor can cheap ones be used in large excess unless a well tried method of recovery of used materials is without question available. While acid processes of extraction may at present be used profitably in treating low-grade vanadium ores, competition may, in the near future, cause valuable property based on these methods to become idle and worthless.

As before stated the greatest difficulty of vanadium production will have been overcome when the vanadium is in solution and free from gangue. The removal of impurities from this point on may in some cases be more expensive per ton of material handled, but the vanadium will be in a condition in which it is readily handled, either by precipitation or otherwise. Specifically I shall discuss only vanadium solutions obtained from the leaching of vanadiferous sandstone, because those interested in solutions obtained by acid leaching will find data bearing directly upon the treatment of such solutions as well.

Nature and Composition of Mill Solutions.

The compounds present in mill solutions will be sodium vanadate and salts of lime, alumina, silica, titanium and sodium, particularly sodium chloride. In addition, some of the vanadium will be combined with each and every other element present, probably including vanadium itself. In neutral solutions there will be a little, and of course in acid solutions there will be a great deal.

In beginning to leach the first solution from the ore will be very low in vanadium, 1.1 g per liter, and will be rich in calcium chloride, calcium carbonate and sodium chloride. This solution is usually colorless and its content depends upon many conditions, such as character of roasting process, fineness of ore, rapidity of leaching, etc. Following the clear liquor will percolate a yellowish solution containing a maximum amount of vanadium, together with sodium chloride and all of the other impurities. Treating one ton of ore with a ton of water, approximately 10 per cent of the solution will be of the first variety, 20 per cent of the second and the remainder which will percolate through will have a decreasing vanadium content following the parabolic law of leaching. The richest liquor should have a vanadium content of about 8 g per liter under favorable conditions. If the liquors are allowed to mix as they come from the ore precipitation of calcium vanadate will occur in the receiving tank. This vanadate of lime usually carries about 18 per cent V_2O_5 and may be filtered and washed.

The greater part of the basic impurities contained in the solution may be removed by the careful addition of a slight excess of sodium carbonate and heating the solution to 90° C. Enough carbonate must be added to make the solution distinctly alkaline. The impurities will settle to the bottom of the tank and the purified solution may be removed by decantation. The residue is of a dirty gray color and contains usually about 10 per cent vanadic acid. While practically all of the lime and alumina may be removed in this manner, the silica usually remains in solution. I have found no certain method of removing the silica and except where products of exceptional

purity are required there is no necessity for removing it. As a matter of fact there is no need of removing any of the impurities unless a very pure product is to be finally extracted from the solution.

In case the solutions are acid with sulphuric acid, the principal impurity will usually be alumina, the other impurities being obviously lime and iron, together with any fixed alkali which may have been present in the ore either originally or added for purposes of extraction. The vanadium is present as vanadyl sulphate, at least 15 different forms of which have been studied, representing the various states of oxidation of both the vanadium and the sulphur. In mill practice the sulphuric acid solutions of vanadium are either green or blue, which colors show a lower state of oxidation of the vanadium than the pentoxide. If to such an impure solution peroxide of hydrogen is added a yellow precipitate is thrown down, which, if filtered off and dried, turns red. Analysis shows it to be a vanadate of iron. This same precipitation may also separate after the solution has stood for several days in an open tank. There is of course some significance to be attached to this reaction, but it is too remote to be readily observed. It is only mentioned here to illustrate the tendency of vanadium compounds to do the unusual thing, as compared with the better known elements.

The removal of the impurities from an acid solution without the addition of alkali in excess is difficult, if not impossible. Sulphate of calcium, insoluble under most conditions, is here held in solution in considerable quantities and may be removed only by an expensive electrolytic method.

If the acid of the solution is hydrochloric, the vanadium is present as vanadyl chloride, in which two atoms of chlorine are united with one vanadyl group. The same impurities are present, lime, of course, being in considerable quantity.

It should not be forgotten that most vanadium ores contain traces of titanium which is extracted with alkali and remains with the vanadium through the various stages of purification, and in some instances the removal of this element may necessitate a modification of the process.

It is of course obvious that it is as necessary to get the vanadium out of solution completely and efficiently as it was to obtain a complete extraction. Not only is it desirable to precipitate all of the vanadium from the solution, but it is necessary that it should be in a chemical combination and physical state suitable either for further refinement or for use by the ultimate consumer. Upon these conditions depends the method of precipitation or whatever refinement of the liquor is necessary.

Since the most important vanadium product is ferrovanadium, the various methods of preparing vanadate of iron, from which ferrovanadium may easily be made, will be discussed.

Preparation of Vanadate of Iron.

The method now in general use for the production of vanadate of iron is to add ferrous sulphate to a neutral or nearly neutral solution of sodium vanadate. The compound is precipitated as either a yellow or dark green, bulky mass, which settles to the bottom of the tank, and from which about 75 per cent of the liquor may be decanted. The usual practice is to filterpress the sludge and dry the cakes in a suitable oven.

There are a number of things about this method which are undesirable. In the first place vanadate of iron, like all other vanadates, is soluble in mineral acids. The first requirement, therefore, is that no free acid be present in the solution or be formed there during the precipitation. The sodium sulphate formed in the solution by substitution is a powerful solvent for the vanadate of iron unless the iron sulphate be present in considerable excess, and even then it is doubtful if a quantitative reaction takes place. If the solution should be slightly alkaline the hydroxide of iron is formed at the same time, thus lowering the grade of the product. Another serious objection to the method depends upon the fact that most of the vanadium in the mill solution is reduced; that is, there are vanadic and hypovanadic salts combined with the sodium after roasting, and

these salts are apparently unchanged in the leaching process. Again, the iron is added in a ferrous condition which does not improve matters. The result is that a small per cent of colloidal precipitate is formed, which makes filtering a difficult problem.

At the same time there are formed certain sulphates of vanadium and iron which are more soluble in water than in the mill solution, so that after the washing has reached a certain stage, provided the precipitate can be washed at all, the vanadium begins to appear in the wash water. This would not be so serious if it were not for the fact that, unless removed by washing, the impurities remaining in the vanadate of iron mean havoc in the electric furnace at the time of final reduction. The vanadate of iron as removed from the filterpress carries in the neighborhood of 90 per cent of the mill solution. It is obvious that the dried, unwashed product must contain an enormous amount of the salts originally contained in the mill solution. These consist mostly of sodium chloride, sodium sulphate, calcium salts, titanium salts and salts of silicon and aluminium. As stated above, some of these salts are in combination with the vanadate of iron and some held mechanically by the sludge. That the presence of these salts is detrimental to electric furnace reduction is easily shown by the heating of the impure product in an open crucible; indeed, it is claimed that a loss of 10 per cent of the vanadium has been known to occur in the simple process of driving off the moisture at a moderate temperature. An instance was cited in which the appalling loss of 70 per cent of the vanadium value was experienced in trying to reduce some vanadate of iron to ferrovanadium.

Unless special precautions have been taken to remove them, silica and alumina as well as lime are always present in the commercial product. Producers of ferrovanadium say that small quantities of these impurities do not affect the value of the vanadate.

Electrolytic Methods of Precipitation.

While it is a well-known fact that salts in solution can be acted upon and changed to desirable forms by the proper application of the electric current, the importance of this in metallurgical operations has not until recently been fully realized. The deterioration of anodes is one of the serious defects of many proposed metallurgical schemes, but there are many instances in which this fact may be used to advantage, and there are processes in operation at the present time in which highly desirable anode products are economically and efficiently turned out.

If two iron plates with a difference of potential of two or three volts are suspended in a receptacle containing vanadium liquor, immediate precipitation of iron vanadate takes place at the anode and continues until all of the vanadium is precipitated. The reaction is efficient and occurs with great rapidity, as will presently be noted.

In case vanadate of iron is to be made from the mill solutions, the acid wash water, if such is used, is led into the same storage tank as was the rich water solution. The water solution is but very slightly alkaline and the mixture is practically neutral, but the fact of its being either slightly acid or alkaline does not seriously affect the electrolytic precipitation. There will separate out and settle to the bottom of the storage tank a yellow vanadium compound containing iron and calcium. This may be allowed to run into the precipitating tank, as lime is not a deleterious impurity in vanadate of iron.

The precipitating is done in wooden tanks 8 ft. long and 4 sq. ft. in cross-section. Each tank is provided with copper busbars running along the top just inside and supported by porcelain insulators. These bars are supplied with current generated at a pressure of from four to five volts. The electrodes are all of iron, although the material of the cathodes may be copper, or, in fact, any other metallic conductor, even carbon. The anodes are made of cast iron, the dimensions

being 18 in. x 18 in. x 0.5 in. They may be made from scrap if desired. The cathodes are also 18 in. square, but are only 1/16 in. thick. Since the thickness is not important, so far as their work is concerned, they should be as thin as possible on account of the weight and expense.

In the electrolytic production of vanadate of iron from mill solutions the phenomena have been studied at some length. It has been observed that the presence of acid radicals other than vanadium aids the precipitation very materially by exerting a solvent action on the anode during electrolysis. In this case a salt of iron is formed which immediately reacts with the sodium vanadate, with the chemical precipitation of vanadate of iron as would be the case if the same iron salt were added from the outside. If there are no other acid radicals than the vanadium present the precipitation occurs more slowly and the precipitate adheres quite firmly to the anodes, necessitating removal either by scraping or violent agitation. The presence of iron oxide on the anode is beneficial because of the ease with which it is attacked by the vanadic acid radical, and in this case the precipitate is the yellow vanadate of iron instead of the green vanadate. A portion of the lime contained in the solution will be precipitated during electrolysis as vanadate of lime, and this is recovered with the iron vanadate. Some of the calcium is precipitated on the cathode, partly combined with vanadium and partly combined with carbonic acid. In all cases of moderate anode density the precipitate is very easily filtered and washed, but if a very high anode density be used the precipitate will be colloidal.

As stated above, the precipitation begins immediately upon the passage of the current, and under favorable conditions proceeds with extreme rapidity. It is possible to precipitate 95 per cent of the vanadium content from a ton of solution carrying 5 g per liter in five minutes with a current of 300-400 amp at a difference of potential of four volts. It is neither necessary nor desirable to remove all of the vanadium from the solution, because the solution which has been electrolyzed is more active as a solvent upon the ore than pure water.

The temperature at which the reaction takes place most efficiently is about 40° C., although good results have been obtained at 0° C. and at the other extreme of 100° C. The distance between the electrodes may be a matter of convenience, although a distance of about 4 in. was used throughout most of my experiments.

The impurities in the vanadate of iron will be sodium chloride, salts of calcium, iron, aluminium, silica and titanium. As a rule any impurity present in the solution will be found in part in the product. If an especially good grade of vanadate is to be made it would be necessary first to purify the solution, which may be done with little expense.

It will be noted that vanadium is always an anode product in hydroelectrolytic operations. Use is made of this property in several methods for producing vanadium compounds besides vanadate of iron. From a sodium vanadate solution the vanadates of almost all of the metals may be made, including vanadate of gold, but not platinum. Copper vanadate is very easily produced, although conditions are much more limited than when precipitating vanadate of iron.

Copper vanadate is either blue or green, and is usually diluted with oxides of copper. It may be filtered and washed in wooden tanks and dried in copper pans. The compound fuses at about 700° C. and becomes very fluid unless a portion of the vanadium is reduced by the action of the graphite of the crucible used for melting. In this case the melt becomes very pasty with the separation of metallic copper. In this connection it is interesting to note that the copper so reduced carries but a very small percentage of vanadium and the vanadium (copper vanadate) carries much carbon. Copper vanadate is the basis of two processes for the production of vanadic acid.

Production of Vanadic Acid.

One of these methods is electrolytic only in the recovery of

the copper to complete the cycle of reactions. I shall describe the method because it affords an opportunity to call attention to some difficulties which may not at first be apparent.

The vanadate of copper is freed as much as possible from solution by washing on canvas filters. It is then placed in suitable containers and treated with sulphuric acid (1-1). The mass turns green at first, then brown and finally red. Upon the addition of the proper amount of acid, which is a matter of experience, the residual vanadic acid is filtered on canvas filters. It may be washed with pure water until the washings show no copper. If an insufficient amount of acid was used the residue will contain copper, while if too much acid was used the recovery of vanadic acid will be small. Not more than 25 per cent of the vanadic acid content of the copper vanadate can be recovered by the first operation.

The solution containing sulphate of copper and vanadyl sulphate is neutralized by passing it over fresh copper vanadate and is then subjected to electrolysis for the recovery of the copper. The copper is precipitated on plates of size suitable for anodes to be used in the precipitating tanks. The solution containing vanadyl sulphate and free sulphuric acid is used for the final decomposition of copper vanadate, and the cycle of reactions is complete. The process is not particularly efficient for several reasons, which, as stated above, are not obvious. The precipitation of copper from copper-vanadyl sulphate solution is not efficient, although perhaps it could be made so, but the mechanical difficulties of handling are great and so numerous that the method is too expensive. The greatest difficulty is in the accumulation of impurities in the vanadyl sulphate solution, the greatest of which is sulphate of calcium. This salt is very soluble in the copper-vanadyl sulphate solutions and soon loads up so that fresh acid has to be used. The process is operative and is actually continuous as outlined, but it is doubtful if it can ever be used commercially.

If an ingot of fused vanadate of copper is attached to a source of electricity so that it becomes the cathode in a simple cell, in which a dilute solution of vanadic acid is the electrolyte, the vanadium tends to pass to the anode, while the copper tends to remain at the cathode in the metallic state. The copper will remain more or less firmly on the cathode, but the vanadic acid in excess of saturation of the electrolyte falls to the bottom of the cell owing to its inability to adhere to the anode. The anode must be inert, preferably platinum. Any mineral acids present in the solution exert a deleterious action and for this reason the copper vanadate should be as pure as possible. Pure water may be used in the cell because the vanadate of copper is soluble enough to allow the action to proceed to some extent, and as the water becomes saturated with vanadic acid the current passes very readily and the precipitation takes place rapidly. As the action in the cell begins spots of metallic copper appear on the surface of the cathode and the solution becomes yellow, due to the dissolved vanadic acid. As soon as the solution becomes saturated a film of the yellow pentoxide appears on the anode and soon the acid begins to settle out of the solution on the anode and on the walls of the cell. The precipitate is of a rich yellow color and settles to the bottom of the cell in a curdy mass, in which condition it may easily be removed by filtration. No washing of the precipitate is necessary, since the solution contains no impurities if a pure copper vanadate was used. The vanadic acid obtained in this way is easily melted and cast into ingots and should assay 100 per cent V_2O_5 . The method is expensive as compared with other electrolytic methods for the production of vanadic acid on account of the large number of operations necessary to prepare the copper vanadate for electrolysis. However, if a pure product is required the method is far cheaper than any method of chemical separation. It must be remembered that vanadic acid may be obtained pure and yet be contaminated during the final operations of getting it into the desired physical state. This is especially true if it is to be melted and cast into ingots.

Any method of producing vanadium compounds by strictly

chemical means is uncertain and expensive. I refer to methods of which the general public, including myself, are aware. Vanadic acid, for example, may be produced chemically in a number of ways, of which I shall mention two or three as examples.

If a dried sample of vanadate of calcium be treated with the proper quantity of hot hydrochloric acid, either concentrated or dilute, there results a separation of the red pentoxide of vanadium, which may be filtered and washed first with hydrochloric acid and finally with pure water. The vanadic acid made in this manner will contain a small quantity of lime salts and traces of any other impurities that were originally present in the calcium vanadate. It will also contain a considerable quantity of hydrochloric acid combined with the vanadium as vanadyl dichloride. The chlorine content as well as the 10 per cent moisture present may be removed by fusing the product in either platinum or fused silica. Depending upon the experience of the chemist a practically pure vanadic acid may be made from vanadate of lime.

Small quantities of an excellent grade of vanadium pentoxide may be made by precipitating the vanadium with a salt of mercury and finally driving off the mercury by heat. The only waste of mercury occurs in the handling of the precipitate, since all of the metal may be recovered as in the practice of gold retorting.

Perhaps the oldest and best known method of making vanadic acid is by first making ammonium metavanadate and driving off the ammonium salts by heat. The operation is conducted by having a concentrated solution of sodium vanadate which has been previously freed from a large percentage of the impurities originally present. To the sodium vanadate solution is added solid ammonium chloride, previously purified, to supersaturation. The white, crystalline ammonium-vanadium salt separates out and may be filtered and washed. If care is used in preparing the solutions the first metavanadate obtained will yield about 97 per cent V_2O_5 upon ignition. By dissolving the ammonium metavanadate in caustic soda and reprecipitating the vanadic acid may finally be obtained practically pure. In this method the expense of the reagents is prohibitive, besides, the reactions are slow and not quantitative.

Vanadic acid can be prepared by the continued boiling of acid solutions containing vanadium salts. The vanadium separates out of the solution in small quantities at various concentrations. Starting with a given amount of vanadium, perhaps a maximum of 10 per cent may finally be recovered by continued boiling a dilute solution. The pentoxide so recovered is not pure and the method apparently has no commercial value.

Vanadic acid may also be prepared by the treatment of dried sodium vanadate with acid, but as in the above instances, the yield is so small as compared with the expense that totally new methods must be devised if vanadic acid is demanded as a commercial product in large quantities.

Author's Electrolytic Method for Vanadic Acid.

I have devised an electrolytic method which is operative and which may be depended upon for economical production of a very high grade oxide. The method consists in taking advantage of the different solubilities of sodium vanadate and vanadic acid in the same bulk of water, and the procedure is as follows:

Sodium vanadate liquor which does not contain chlorides is purified by heating to 90° C. and making slightly alkaline with sodium carbonate. This liquor is decanted into an evaporator and concentrated to 30° Baumé. At this degree of concentration it will contain about 200 g per liter vanadic acid, depending upon the amount of sodium salts present. This concentrated solution is electrolyzed in a compartment cell consisting of a porous cup with a capacity of 8 liters inside a suitable jar or tank with a capacity of 5-10 gal. The vanadium liquor is poured into the porous compartment and water containing a little lye is poured into the outer compartment. The anode is of platinum foil 6 in. x 6 in., and the cathode is of either iron or copper, preferably the latter. A difference of

potential of 6-8 volts, depending upon the resistance of the porous cell, is taken from a low-voltage generator. A current density of about 30 amp per square foot at the anode will result. The sodium ions pass through the porous cell into the cathode compartment, leaving the vanadium ions in the anode compartment. Since sodium vanadate is very soluble in water, and vanadic acid is almost insoluble in water, a precipitation results upon the removal of the alkali. The presence of mineral acids interferes seriously with this reaction, since the vanadium is soluble in dilute solutions of mineral acids. If a small amount of chlorine is present it is evolved partly at the anode and that which is not evolved causes the formation of various complex compounds which result in a very low grade product. The presence of sulphuric acid in the anode compartment causes the precipitation of sodium vanadyl sulphate containing 73 per cent V_2O_5 .

Any silica or lime salts not removed by the preliminary purification treatment will precipitate in part in the anode compartment just as the solution becomes neutral and may be removed by filtration. In spite of every precaution some impurities will find their way into the product, and not a little of this comes from the disintegration of the porous cell. The greatest objection to this process is the necessity for using platinum in such large quantities, one ounce of platinum being required for every pound of vanadic acid produced per day of 24 hours. There is no loss of platinum, however, and it is probable that platinum plate electrodes would greatly reduce the actual weight of platinum required.

The vanadium pentoxide produced by this method is of the red variety and presents a tough leathery appearance, and may be filtered and washed with ease. Without taking more than the usual precautions, the grade should be 98 per cent V_2O_5 , the impurities being principally lime and silica. The current efficiency is as high as could be expected with a compartment cell. There is no apparent reason why the units may not be of any desired capacity, and it is very probable that the process could be made continuous.

From the foregoing statement, that chlorine interferes with the recovery of the vanadium pentoxide, it is obvious that the salt roast is out of the question. Therefore if vanadic acid is the required product another method of extraction from the ore will have to be used. A modification of the sodium carbonate roast will be found satisfactory, although the extraction will not be so high, nor will the operation be as cheaply performed. A small amount of chlorine will be present even in this case because the commercial sodium carbonate carries about 1 per cent sodium chloride and the commercial sodium hydrate sometimes carries as high as 3 per cent.

Presence of Titanium in Vanadium Ores and Products.

Since titanium is present in almost all vanadium ores, although the percentage is usually very small, its extraction is excellent. This element resembles vanadium in a large number of reactions, and it is natural to expect that titanium should be found as an impurity in commercial vanadium products. I have found titanium in vanadic acid of supposedly high purity, and in some cases the percentage has been higher than one would suppose. The fixed alkalis are good solvents for titanium and this element may in some cases interfere with assays for vanadium. From alkaline or neutral solutions lead acetate precipitates lead titanate, which exactly resembles lead vanadate excepting that the precipitation is not quantitative under the same conditions that lead vanadate is. This precipitate is soluble in nitric acid and in sulphuric acid solution is apparently reduced to sulphur dioxide to the same extent that iron would be if present. The orange red color of titanium with hydrogen peroxide in nitric acid solution may in some cases so nearly resemble the vanadium color that the chemist is at a loss to understand why he cannot precipitate all of the vanadium out of the solution.

In connection with the presence of titanium in vanadium

products, the phenomenon of the evanescent end point when titrating with potassium permanganate may be due to this element.

Vanadium for Steel Making.

Since practically all of the vanadium produced at the present time is finally to be used in making vanadium iron and steels, it is proper to say a word regarding the preparation of the vanadium for use by the steel maker. The vanadium must be added in a metallic form because a portion of its value lies in its property of removing traces of oxygen and nitrogen and uniting with carbon to form carbides in the metal. Any vanadium present as oxide will probably be totally lost in the slags. The form in which vanadium is now added to the steel is an alloy of vanadium and iron carrying from 25 per cent to 50 per cent metallic V. There are difficulties in the production of ferrovanadium which at the present time have not been overcome.

Ferrovanadium may be produced from either vanadate of iron or vanadate of calcium (and of course vanadic acid) by reduction with carbon at a temperature of about 1500°C . The reduction is most efficiently carried on in an electric furnace of any type that will give the proper temperature. The alloy, which melts between 1300°C . and 1400°C ., depending upon the percentage of vanadium, settles easily to the bottom of the crucible, and may be tapped and cast into ingots. Ferrovanadium made in this way will contain a high carbon content, some silicon and aluminium. Steel makers do not want ferro with carbon over 1 per cent at the outside, while the alloy obtained by the above method will contain much more than this.

A passable grade of ferro is being made by reduction with metallic aluminium. The process is necessarily expensive on account of the price of the reducing agent and certain losses not incident to other processes.

Although I have never seen the process in operation, the reduction of vanadium by metallic silicon or ferrosilicon appears to me to be one of the best yet devised. Mr. F. M. Becket, of Niagara Falls, is the inventor of the process. Mr. Becket was the first man in the United States to produce vanadium alloys in commercial quantities, and he probably knows more about this end of the game than any one else. Therefore I have no hesitation in accepting his statements relating to his process, which is based upon sound metallurgical principles.

A recent process patented by Mr. Boericke ought to give a very high-grade alloy and should be efficient. Mr. Boericke uses either metallic manganese or ferromanganese as a reducing agent, thus avoiding the presence of carbon, silicon or aluminium in the final product.

Numerous other processes have been patented, but none appear to be of any considerable importance.

To sum up, it may be said that great advances have been made during the last few years in every branch of vanadium metallurgy. Efficient processes of extraction, precipitation and producing alloys have been developed and these, at the present prices of vanadium products, allow the various ores to be worked at a profit. As with problems concerning other metals, none of the processes are entirely satisfactory in the fullest sense. Owing to the fact that vanadium seems to be firmly established in the steel industry, and promises to be of great value in other lines as well, and since competition is going to force the prices steadily downward, one is compelled to predict that processes of extraction and refining of high efficiency will be developed in the near future. The time when operators can slop a little acid on picked ore and make a profitable extraction, or throw away 1 per cent roasted tailings, or gracefully accept a penalty for uranium values in vanadium products will soon be past, and so much the better for metallurgical intelligence and really careful experiment.

I wish to thank my assistant, Mr. Arnold J. Guerber, for months of painstaking work on the problems here discussed and for carefully prepared analytical data upon which many

of the conclusions are based. In papers of this character, because of the newness of the subject, there must have crept in some misstatements and hasty conclusions. I shall be very glad to have attention called to those which may be observed.
Boulder, Col.

The Electric Pig Iron and Steel Plant at Trollhättan, Sweden.

In the July issue, page 368, and in the September issue, page 459, the first parts were published of the important paper presented by Mr. J. A. LEFFLER to the Jernkontoret (the Swedish Association of Iron Masters) at their meeting at Stockholm on May 31 on the electric iron ore reduction plant at Trollhättan, Sweden. In the following we give an abstract of the concluding portion of Mr. Leffler's paper, together with an abstract of a paper by Mr. E. ODELBORG on the conversion of the electrically reduced pig iron into steel.

Electrodes and Electrode Consumption.

Part of the electrodes used at the works were supplied by the Planiawerke, Ratibor, Silesia, and a part by the Höganäs-

especially the percentage of sulphur should be kept as low as possible. With the electrode consumption found in actual operation and assuming that all of the sulphur combines with the iron, 1 per cent sulphur in the electrodes corresponds to 0.005 per cent to 0.006 per cent sulphur in the iron.

The electrodes in the furnace are four in number. Each electrode is built up of 4 cartons, 2 m long and 330 x 330 mm in cross-section arranged to form an electrode of 660 x 660 mm cross-section. Before being bound together the sides forming the joints are planed to a flat surface and the joint is made with a thick paste of molasses and graphite, so that the closest joint possible is formed. The upper part of the electrode is covered with asbestos paper and thin sheet iron and the top surface is covered with a thick layer of ground asbestos and silicate of potash.

The sides of the electrodes are carefully planed smooth at the upper part for a distance of about 250 mm in order to make a good contact surface for the electric contacts. Between these and the electrode is also placed a fine mesh copper wire cloth.

In order to observe the quality of the different makes, electrodes were built up of carbons from the two factories and



FIG. 1.—FURNACE HOUSE AT TROLLHÄTTAN.

Billesholm Company, Sweden. The purpose in using both was to obtain a comparison between the well known Planiawerke electrodes and electrodes of Swedish manufacture. Results of the analyses of the electrodes are shown in Table I:

Analysis of Electrodes.

	Electrodes from	
	Plania.	Höganäs.
Ash	2.80%	3.96%
S (total)	0.79%	1.06%
P	0.27%	0.16%
Composition of the Ash:		
SiO ₂	0.44%	0.97%
P ₂ O ₅	0.37%	0.627%
CaO	42.00%	42.00%
Na ₂ O + Na ₂ CO ₃	1.21%	0.45%
FeO	6.08%	10.20%
MgO	2.62%	2.16%
Al ₂ O ₃	28.04%	21.70%
Mn ₂ O ₃	0.52%	0.38%
Fe ₂ O ₃	21.22%	19.80%
Total	98.557%	98.03%

The rather large difference in the ash analyses is probably due to the difficulty to burn the last traces of carbon by ignition.

From the above will be seen that the Höganäs electrodes contain a somewhat high percentage of ash and sulphur. In producing iron of high quality it is, of course, necessary that

each kind used under similar conditions. No difference in durability could be discovered.

Table II gives the complete record for two electrodes (called No. 1 and No. 2), also the average for ten electrodes.

Electrode.	Working Time.		Weight in Kg.		Consumption of Electrode.		Consumption per Ton of Iron.	
	Hours.	Min.	Before.	After.	Total Kg.	Per Cent.	Gross.	Net.
No. 1	1041	37	1321	557.5	763.5	57.79	2.09	1.21
No. 2	1115	31	1325	743	582	43.92	1.85	0.81
Average of 10	906	6	1313.3	635.8	677.5	51.61	2.28	1.16

The Charges.

During the time which this report concerns 29 different qualities of charge were used, which, however, may be regarded as four main groups, viz. unroasted Tuollavaara, roasted Tuollavaara, Borgvik and Uddeholm ores.

Referring to Table III it will be noticed that the variations in the first group mainly concerned modifications of the silica contents of the slag and also varying proportions of Tuollavaara ore in the charge.

In the third group a complete change was made in the mixture of the ore at the request of the iron works from which

TABLE III—(Continued)

Per cent. Iron in		Slag per Ton Iron		Material Charged per 100 l.		Charcoal per Ton Iron						Electric Power						Consumption of Electrodes					
						Containing						Time Used		Average Load		Kw-hours Used		Iron Produced per Kwh. Year	Gross	Net	Pr Ton Iron		
Ore	Charge	kg	kg	cu. m.	Water	Gases	Ash	C Coke	Total	For Working	For Interruptions	Total	Hours	Min.	Hours	Min.	kw	Total	Pr Ton Iron	kg			
Group I	62.10	205	66.49	2,422	69.1	41.7	11.8	293.1	415.7	2,009	56	105	39	2,115	35	1,319	2,651,029	2,296	3.82	13,012	6,743	11.24	5.83
Group II	62.56	224	71.13	2,247	50.8	36.9	11.0	277.6	376.3	184	32	4	58	189	30	1,694	312,601	2,149	4.08	1,578	763	10.84	5.24
Group III	42.42	780	90.31	2,610	59.8	49.3	13.2	323.4	445.7	639	18	20	57	660	15	1,017	650,480	2,623	3.34	2,281	1,121	9.19	4.52
Group IV	53.06	458	69.88	2,697	40.2	43.1	17.2	325.7	426.2	506	34	22	11	528	45	1,733	877,706	2,643	3.31	2,474	1,285	7.45	3.87
Grand Total	57.00	327	70.77	2,479	60.9	42.9	12.8	301.4	418.0	3,348	10	153	45	3,501	55	1,344	4,500,596	2,391	3.66	19,345	9,912	10.28	45.27

it was sent and a gradual increase of lime was necessitated on account of the sulphur contents of the iron.

The fourth group shows a gradually increasing percentage of concentrates from Persberg which was mixed with the ore.

Table III forms a summary report of the various quantities of ore, charcoal, energy and electrodes used for respective charges and also the amounts of pig iron and slag produced from four groups of charges.

The Durability of the Crucible and Roof.

The roof of the crucible was at first considered as being a weak point of the construction, but it may be stated that it has stood the test in an excellent manner, as only two repairs of any importance have been necessary.

In the neighborhood of the electrodes the arch became red hot in a few places after protracted working, but it generally proved possible to prevent the bricks from burning through by cooling them with an air blast.

It is probable that the roof would have lasted still better if the furnace could have been worked the entire time with both phases. The second phase was, however, at times out of action. In this way the heating of the roof became uneven and it was damaged by cracks resulting from the stresses.

In the morning of April 6 when the charge contained 70 per cent concentrates a settling occurred from the shaft into the crucible which was accompanied by so violent a development of gas that the roof was damaged between two electrodes. At the time the loose bricks closing the holes in the roof were so tightly sintered into their places that they did not act as safety valves. Repairs were, however, effected in less than two hours. Once the iron cut through the lining of the hearth and the iron shell, causing an interruption of about two hours.

Iron and Slag.

Analysis of the iron was made from each tapping and analysis of the slag and the gas at very frequent intervals. The accompanying tables IV and V show four different analyses which may be taken as averages for those taken and the average of all the analyses are also given. The complete tables are of great interest as showing the continuous and regular working of the furnace, but are too extensive to publish in these columns.

TABLE IV.
Analyses of Iron.

Date.	Tapping.	Composition in Per Cent.				
		C	Si	Mn	S	P
Jan. 3, 1911	No. 138	4.19	1.35	0.90	0.004	0.021
Jan. 14, 1911	No. 171	4.04	0.75	0.82	0.005	0.020
Jan. 16, 1911	No. 324	3.0	0.14	0.08	0.028	0.019
Jan. 30, 1911	No. 372	3.1	0.45	0.40	0.014	0.010
Min.		2.30	0.03	0.08	0.004	0.010
Max.		4.50	2.21	1.14	0.094	0.026
Average		3.2	0.3	0.3	0.028	0.015

TABLE V.
Analyses of Slag.

Date.	Tapping.	Composition in Per Cent.									
		SiO ₂	Al ₂ O ₃	TiO ₂	FeO	MnO	CaO	MgO	CaS	P ₂ O ₅	Silicon*
Jan. 3, 1911	138	44.76	1.20	2.58	1.75	1.35	31.70	15.10	0.077	Trace	1.61
Jan. 14, 1911	171	41.60	6.85	2.72	1.49	1.48	28.91	16.70	0.063	1.70
Jan. 16, 1911	324	46.82	5.06	6.89	0.23	33.24	7.97	0.023	0.041	1.91
Jan. 30, 1911	372	37.98	6.98	0.37	1.28	0.52	27.98	23.45	0.123	1.42
Min.		36.40	1.20	0.36	0.50	0.08	25.00	3.55	0.014	1.31
Max.		49.20	12.85	5.92	10.92	4.48	35.00	23.45	0.217	1.94
Average		43.00	5.00	3.00	2.00	2.00	30.00	15.00	0.050	1.60

* SiO₂, TiO₂, Al₂O₃

FeO, MnO, CaO, MgO

A subject of great interest is the question whether by this process iron of a uniform composition can be produced for any desired length of time.

As no O and C, except what is contained in the ore, the fuel, and the electrodes, is introduced into the furnace, it is evident that the process is to a large degree dependent on the relative proportions between the quantities of O and C so introduced. It is also apparent that these proportions are apt to vary owing to variations of the fuel and ore and the gas circulation. These conditions must be watched carefully and the charges proportioned accordingly. Irregularities of work-

*Net—51.24% of gross.

ing can also be remedied by additions of ore and charcoal through the openings in the roof of the crucible.

The problem of even gas circulation cannot yet be considered satisfactorily solved. It is apparent that the CO_2 percentage of the escaping gas is largely dependent upon the gas circulation and for same reason if the latter is disturbed, as for instance by choking charcoal or ore dust in pipes, etc., the CO_2 percentage will change with consequent effect in the supply of C.

In studying the iron and slag analysis these facts should be taken into consideration and the results appear better.

The analyses of slag show that these have been kept more siliceous than desirable for the basic lining of the hearth. This has, however, been done for the purpose of obtaining results as closely comparable as possible with the treatment of the same ores by the ordinary blast-furnace process.

A few observations of the temperatures of iron and slag taken by means of an optical pyrometer are given in Table IX. In the table is also shown the percentage of Si and Mn in order to show the connections between these and the temperatures.

Composition and Pressure, etc., of the Furnace Gas.

Throughout the entire time of working the gas has been the object of careful observation through analyses and determinations of speed and calorific value.

During the time to which the report refers over 1,800 analyses have been made of gas collected from different portions of the furnace. A large portion of these are complete, but in the majority of cases only the CO_2 was determined. A number of direct determinations by calorimeter of the thermal value of the gas were also made.

Tapping.	Per Cent.		Temp. in deg. Cent.	
	Si.	Mn.	Iron.	Slag.
169	0.80	0.76	1380	1420
174	0.81	0.83	1330	1375
225	0.14	0.26	1270	1290
401	0.13	0.12	1280	1370
Min.	0.04	0.06	1270	1270
Max.	1.26	0.77	1380	1470
Aver.	0.40	0.30	1300	1376

Date, 1911.	Composition in Per Cent of Volume.						Dry Gas.	
	CO_2	O	CO	H	CH_4	N	Weight in kg. per Cu. M.	Heat Val. ue per Cu. M.
Jan. 3	28.2	1.276	2113
Jan. 14	27.2	..	57.5	14.8	..	0.5	1.302	2112
Jan. 16	24.4	..	56.4	10.0	1.8	7.4	1.288	2239
Jan. 30	21.8	..	65.4	10.4	..	2.4	1.171	2018
Min.	3.2	0	46.2	2.5	0.8	0.4	1.377	3030
Max.	31.2	1.2	82.0	20.2	9.1	14.3	1.225	2500
Aver.	23.0	0	60.0	10.0	2	5		

In spite of the large number of analyses made it would be premature to draw any definite conclusions concerning the action of the gas under different conditions. It is, however, apparent that currents of varying composition are apt to pass through different parts of the shaft so that, for instance, the gas at the centre of the furnace and at the side in the same section may show a different composition.

The outside temperature of the furnace was kept constant and normal but in a couple of cases when some damage was done by overheating.

An average of 110 liters per minute of cooling water was used for cooling the electrodes with a loss of an average of 97.8 kw, which is nearly 7 per cent of the total load on the furnace.

Working Results.

While it may not be advisable to draw final conclusions on the basis of the results obtained it is nevertheless of interest to examine these results with a view to find indications for the continued development and improvement of the process.

As a definite and highly satisfactory experience it should be noted that the need of repairs of the crucible has been very slight. During a period of nearly 5 months of continuous working only about 18 hours have been required for repairs which must be regarded as an eminently satisfactory result.

It is also satisfactory to note that the height and other dimensions of the shaft have in the main proved correct for normal conditions for fuel and ore.

If, however, earthy ore or concentrates or too fine fuel is used, or if the charge contains too much moisture the working is injuriously affected as would equally be the case with an ordinary blast furnace, and the design of the shaft does not appear to be well suited for such conditions. This is because it is then especially important to maintain a strong circulation of gas.

If this cannot be done the moisture is not evaporated, but gradually penetrates further down in the shaft causing a lowering of the temperature. The reducing power of the CO is then decreased and the percentage of CO_2 in the resulting gas decreases. But this percentage of CO_2 is the indicator of the degree of utilization of the fuel used and must, therefore, under all circumstances be maintained as high as possible.

For the conditions mentioned a shaft with a wider neck would be more suitable, as it leaves a larger space for the gas to pass up and a reduction of the temperature would be counteracted. Possibly it would also be an advantage to make the shaft wider and lower.

In order to produce an effective and uniform circulation of the gas it is in the first place necessary to purify it by washing, so that the frequent interruptions for cleaning may be avoided. Further the gas pipes must be quite tight so that the gas drawn by the fan is really blown into the crucible, which must also be made as tight as possible. Alterations in these respects are at present in progress at the works.

The circulation of gas which is so useful and necessary for the purpose indicated above and which moreover is a protection for the roof, has, however, the disadvantage that the electrodes are attacked by the carbonic acid in the gas. Nevertheless, the electrode consumption has been low. The average for the entire time per ton of iron has been 10.28 kilogrammes gross and 5.27 kilogrammes net. Electrodes with screw joints are now being adapted for the furnace and the gross consumption may then be reduced—under most favorable circumstances to equal the net consumption.

The experience with this furnace appears to indicate that with this size four electrodes is a minimum number. It would probably be better to use six electrodes with three-phase currents. This would distribute the heat more uniformly and the risk of disturbances of the work on account of defects in the transformers would be reduced. Losses through cooling water would in this case probably be increased, but this would undoubtedly be more than counterbalanced by the advantages gained.

During the time to which the report refers only unburnt limestone has been used. In view of future comparisons with ordinary blast furnace results it was desired to maintain the conditions very much similar to those under which the same ores are usually treated. It is, however, evident that by the use of burnt lime the economy of the process will be improved and this improvement would increase in proportion as the quantity of lime required is increased. Whether unburnt or burnt limestone is used the percentage of CO_2 in the gas remains the same. This proves that the CO_2 derived from the limestone may be replaced by CO_2 formed by the reduction of the ore by CO. The power required for burning the limestone is, therefore, saved and the same quantity of fuel reduced more ore.

* * *

While Mr. J. A. Leffler's paper dealt with the reduction of pig iron from iron ore in the electric furnace at Trollhättan, Mr. E. ODELBORG presented a paper on the results obtained in the conversion of this electrically produced pig iron into steel. An abstract of Mr. Odelberg's paper is herewith given.

As the manufacture has been in progress only for a short time, and as further the appearance and analysis of the pig iron were in some respects not the best that could have been desired during the first period of working, it was with some hesitation that the steel works received and treated the product.

The results up to the present time are, therefore, few and of indifferent interest and perhaps somewhat one-sided.

The Behavior and Qualities of the Electric Pig Iron in the Open-Hearth Process.

For acid furnaces it is desirable to have a pig iron with 0.4 to 1.0 per cent. Si or more, and many manufacturers consider that they require at least the same quantity of Mn to obtain a fully satisfactory product.

At the Trollhättan plant there was at first produced a considerable quantity of iron which as regards surface and fracture had the characteristic appearance of spongy iron and of which the analysis showed only a very small fraction of Si and Mn. At that time there was, therefore, some hesitation about the treatment of this iron, the material for which was the same as ordinarily used for the production of a high-grade steel.

The first test of using this iron at Degerfors was made in a basic furnace with addition of ordinary grey iron and a smaller quantity of scrap than usual in order to make sure of an acceptable product of hot iron. The result was, however, so satisfactory that already at the third charge only white electric pig iron was used, together with 50 per cent. scrap. The somewhat surprising observation was then made that the expected boil of the material immediately after the melting did not occur.

The tests with melting of spongy pig iron were then transferred to an acid furnace. Small quantities were used at first but these were very soon increased to the ordinary proportions of 64 per cent. pig iron and 36 per cent. scrap.

As anyone acquainted with the open-hearth process is aware there occurs immediately after, and even during the melting, a violent boil of the material. This is especially the case in an acid furnace if iron too poor in silicon is used (even if this is not what is called spongy iron). A similar partial boil occurs again on teeming the charge into the ladle and moulds and this is invariably the case with hard steel. It was, therefore, surprising that this partial boil did not occur, but instead the regular "boil" commenced immediately after the material was melted and before the oreing.

The charges of such iron were finished in considerably shorter time than the ordinary (about 8 hours). They also required considerably less oreing, but nevertheless were quite hot and liquid when teemed. With equal additions of Mn and Si as ordinarily a product was obtained with the same percentage of Mn and Si as is generally obtained with pig iron of normal analysis from an ordinary blast furnace.

When, however, the same open-hearth furnace was charged with electric pig iron of correct appearance and normal percentages of Mn and Si, together with the same proportion of scrap as before the treatment required considerably longer time and more ore than can be regarded as desirable.

These conditions are elucidated in the working results of Degerfors which show that the time and oreing of the charges vary generally in accordance with quite small variations of the Si and Mn contents in the ore.

At Degerfors a decided opinion was soon arrived at that electric pig iron of spongy nature was that most suitable for the open-hearth process, whereas the refining of pig iron of normal fracture and analysis proved costly and requires too much time. In regard to the finished product from these apparently so different qualities of pig iron no difference has been found at Degerfors either in teeming or in use.

On the strength of this experience and as on account of decreased output it proved more difficult and costly to produce in the electric reduction furnace pig iron of so-called normal silicon contents, it was decided that the manufacture should be arranged for producing spongy pig iron and that there should be no hesitation about producing the spongy quality, which it is considered desirable to avoid in ordinary blast furnace practice.

Various explanations have been put forward to explain the cause of the rather surprising result of the refining of electric pig iron, which in appearance as well as composition, has been the opposite of pig iron produced in the ordinary blast furnace.

The true explanation is probably that the electric pig iron, even of the white spongy quality, is practically free from rests of ore and iron reoxidized by the blast. This is also proven by the fact that the fracture of very thick flowing spongy iron from the electric furnace is perfectly white and free from tarnished bubbles, while the bubbles in spongy iron from the ordinary blast furnace usually show all the colors of the rainbow. On account of this freedom from oxygen the partial boil in the open-hearth furnace during and after the melting does not occur and it is possible to at once decarburize the charge by oreing. The presence of silicon in such pig iron is therefore unnecessary during the first portion of the open-hearth process.

The slowness of the open-hearth process with electric pig iron of normal appearance is due to the silicon not being oxidized at once by oxides in the iron, so that the refining must be accomplished principally by means of ore and only after this is accomplished the decarburizing proper can commence.

In proof of the above it may be mentioned that for a charge of pig iron with 0.61 per cent. Si and 0.68 per cent. Mn a time of 50 minutes was required to reduce the Si and Mn to the low percentages necessary for beginning the decarburization. If the Si and Mn contents are reduced below these limits in an acid furnace, it would appear that they are again absorbed and not again reduced during the decarburization. The carbon contents in the pig iron used for the charge then determines the time and the quantity of ore required for the decarburization to obtain the desired product. As a higher, and according to the ordinary rules, normal percentage of silicon cannot occur in spongy iron from an electric reduction furnace, the Si contents alone has been referred to in this connection as determining the quality of the pig iron. But this iron is also distinguished by being so low in carbon as to approach steel and it should be remembered that the higher percentage of carbon which occurs at the same time with a higher percentage of silicon also has the effect of prolonging the time for decarburizing in the open-hearth furnace.

The charge above referred to, the pig iron for which was mottled and contained 3.69 per cent. C, 0.61 per cent. Si and 0.68 per cent. Mn, required a time of 6 h. 25 min. for a product with 0.60 per cent. C. Another charge for which the pig iron (spongy iron) held 2.75 per cent. C, 0.04 per cent. Si and 0.19 per cent. Mn required, however, a time of only 4 hours for yielding a product with 0.45 per cent. C. In these times mentioned the time for cleaning and repairing the hearth is not included.

A higher percentage of silicon than 1 per cent. in pig iron from an electric reduction furnace is, therefore, if not detrimental to the quality of the final product, an obstruction to the progress of the open-hearth process, and to a certain extent the presence of manganese in such iron has a similar effect.

It is well known that a certain amount of silicon (many manufacturers also demand a considerable percentage of manganese) is necessary in pig iron from an ordinary blast furnace in order to make the iron suitable for the open-hearth process. It is also well known that if the Si and Mn contents in the iron are too high the open-hearth process is protracted, which is undesirable from an economical point of view. The quantities of Si and Mn required for carrying through the process with pig iron from an ordinary blast furnace, are just those needed during and immediately after the melting of the charge to free the iron from the oxides and gases formed by the influence of the blast. Whatever surplus of Si and Mn may be present only protracts the open-hearth process unnecessarily.

The partial boil frequently mentioned in this connection is a premature and gradually increasing partial decarburization of detrimental effect, but it does not occur if the pig iron contains sufficiently large quantities of Si and Mn. The fact that it does not occur in the case of "electric" pig iron with practically no Si and Mn must be explained by this pig iron being free from oxides.

It may therefore be maintained on the strength of the experience gained with electric pig iron treated by the open-hearth process that high silicon contents are detrimental rather than advantageous, while with ordinary pig iron a certain quantity is necessary to neutralize the defects of a reduction process less perfect and ideal than the electric process.

Steel Produced from Electric Pig Iron.

As already mentioned, the steel made at Degerfors, whether produced from pig iron very poor in silicon or from what may be called more normal pig iron from the Trollhättan furnace, has always proved of very good quality when teemed. Such has also always been the case on rolling as indicated by forging and annealing tests as well as by the analyses.

The first product which was rolled to wire was sent to the Blombacka A. B., Lindfors, where tests were immediately made and the quality found to be very good in every respect.

Garpshtyte Fabriks, A. B., have also tested a few small lots of rolled wire and obtained very satisfactory results.

A lot of 20 tons with 0.75 per cent. C was sent to one of Degerfors's largest customers for rolled wire for testing. They wrote as follows:

"We have tested the 0.75 per cent. C material you sent us and found that the wire is perfectly satisfactory, but we have not found any particular difference between this material and your usual quality with 0.75 per cent. C which we are in the habit of buying. Kindly note that we have no objections against the electric steel with 0.75 per cent. C. It is very satisfactory, but we cannot say that it is superior to your usual 0.75 per cent. steel."

A quantity of files and axes have been made from steel with 1.25 per cent. C and 0.55 per cent. Mn. The files are in use at Degerfors and have proved to be of good quality. The opinion of the file makers in regard to the behavior of the steel in working was that it was soft and good both for forging and cutting, but required very little annealing, although the carbon was as high as 1.30 per cent., and they were of opinion that 1.40 per cent. C would probably have been more suitable.

Fuel-Oil Specifications.

In consequence of the success that has attended the use of heating-value specifications in buying and selling coal there has arisen a demand for similar specifications, based on careful tests, to govern the purchase of fuel oils and the refined products of petroleum.

The Bureau of Mines has therefore issued Technical Paper No. 3, by Irving C. Allen, on "Specifications for the purchase of fuel oil for the Government," with directions for sampling oil and natural gas.

"These specifications, to be satisfactory," says the author, "should establish not only the heating value of the oil, and thus show, like the specifications for coal, the number of heat units the purchaser obtains for a given price per unit quantity of fuel, but also the physical character of the oil, its flash point and burning point and the quantity of extraneous matter it contains.

"It is evident that an essential feature of any such plan of purchasing oil in bulk is an equitable method of sampling deliveries—that is, a method that insures the taking of representative samples.

"The Bureau of Mines has received many inquiries from Government bureaus and from private concerns regarding fuel-oil specifications and the sampling of purchases. In

reply to these inquiries the bureau issues this paper, which, although intended primarily for the guidance of Government officials, may be of service, the bureau trusts, to all persons who buy or sell fuel oil."

General specifications for the purchase of fuel oil are given as follows:

In determining the award of a contract, consideration will be given to the quality of the fuel offered by the bidders, as well as the price, and should it appear to be to the best interest of the Government to award a contract at a higher price than that named in the lowest bid or bids received, the contract will be so awarded.

Fuel oil should be either a natural homogeneous oil or a homogeneous residue from a natural oil; if the latter, all constituents having a low flash point should have been removed by distillation; it should not be composed of a light oil and a heavy residue mixed in such proportions as to give the density desired.

It should not have been distilled at a temperature high enough to burn it, nor at a temperature so high that flecks of carbonaceous matter began to separate.

It should not flash below 60° C. (140° F.) in a closed Abel-Pensky or Pensky-Martens tester.

Its specific gravity should range from 0.85 to 0.96 at 15° C. (59° F.); the oil should be rejected if its specific gravity is above 0.97 at that temperature.

It should be mobile, free from solid or semi-solid bodies, and should flow readily, at ordinary atmospheric temperatures and under a head of 1 foot of oil through a 4-in. pipe 10 ft. in length.

It should not congeal nor become too sluggish to flow at 0° C. (32° F.).

It should have a calorific value of not less than 10,000 calories per gram (18,000 British thermal units per pound); 10,250 calories to be the standard. A bonus is to be paid or a penalty deducted according to the method stated under section 21, as the fuel oil delivered is above or below this standard.

It should be rejected if it contains more than 2 per cent water.

It should be rejected if it contains more than one per cent sulphur.

It should not contain more than a trace of sand, clay or dirt.

Each bidder must submit an accurate statement regarding the fuel oil he proposes to furnish. This statement should show: The commercial name of the oil; the name or designation of the field from which the oil is obtained; whether the oil is a crude oil, a refinery residue, or a distillate; the name and the location of the refinery, if the oil has been refined at all.

The fuel oil is to be delivered f.o.b. cars or vessel, according to the manner of shipment, at such places, at such times, and in such quantities as may be required.

Should the contractor, for any reason, fail to comply with a written order to make delivery, the Government is to be at liberty to buy oil in the open market and charge against the contractor any excess of price, above the contract price, of the fuel oil so purchased.

Copies of this Technical Paper may be obtained by addressing, the Director of the Bureau of Mines, Washington, D. C.

Regulations on Papers for the Eighth International Congress of Applied Chemistry, 1912.

In the following we give the set of tentative rules on papers to be presented before the Eighth International Congress of Applied Chemistry. The Congress will be held in this country in 1912. Since the final rules are to be adopted during December of this year, any criticism of the tentative rules given

below should be made before Dec. 1, 1911, and should be sent to the secretary of the Congress, Dr. B. C. Hesse, 25 Broad Street, New York.

1. All papers must be in duplicate and legibly written, preferably typewritten.

2. Each sheet must be written on one side only and *not* on both sides.

3. Each paper must be accompanied by an abstract thereof, also legibly written, preferably typewritten, and which must be in duplicate.

4. Papers and their abstracts, both in duplicate, must be in the hands of the American Committee not later than June 30, 1912. All papers received prior to that time and accepted will be printed in their respective Sectional Volumes and distributed to such of the attending members of the Congress as may desire them at or before the opening of the Congress. Papers received after that time, if accepted, will be printed, but may appear in an appendix which may or may not be ready by the opening of the Congress; the Congress cannot then undertake to print them along with the papers of those sections to which they may be assigned.

5. All papers or like contributions must be as concise as possible, and must contain the full name and post-office address of their respective authors; further, what number, if any, of reprints of the paper or like contribution is desired.

6. Papers or other like contributions must be original and not elsewhere read or published, nor contributed or offered to any other society, association or publication for presentation or publication. The offer of any paper to the Congress is a tacit and understood agreement to the foregoing requirement. No paper should deal with historical matter any more than is needful for a proper understanding of the new subject matter presented, which subject matter, as far as practicable, should be of a date subsequent to June, 1909, the date of the Seventh International Congress of Applied Chemistry, except by special request. Nonconformity to this requirement may be a reason for rejection; a remedy herefor is offered in paragraph 18.

7. All authors, as a matter of course, agree not to publish their accepted papers in any other publication except as hereinafter provided, and, further, they automatically agree to abide by any final decision of the Congress with respect to such paper or papers, their presentation, discussion or printing.

8. The Congress obligates itself to have its final Report and Proceedings, including subject and authors' index, completed and ready for distribution on or before Dec. 31, 1912; in case those reports and proceedings be not ready for distribution by that date, authors of all papers received and accepted after June 30, 1912, may then publish in any journal or publication that they may elect. This refers only to the report and proceedings bound in paper; members desiring cloth-bound sets can obtain them at an advanced charge over the \$5.00 membership fee; such advanced charge will be announced later, but will probably be \$2.50; delivery of these cloth-bound sets will be about 90 days later than of the paper-bound sets. Authors of papers received before the close of June 30, 1912, may publish those papers in any publication they may elect after the paper is read or after the Congress has adjourned.

9. Authors of papers accepted and printed in full or in abstract will receive free of cost and all delivery charges, not to exceed fifty (50) reprints of each paper or abstract; additional copies of reprints can be had upon payment of the prices for such copies, which prices will be announced later. The Congress cannot undertake to furnish reprints of papers if the order for such reprints is not attached to the paper or abstract when received by the American Committee.

10. No paper offered to and accepted by this Congress can be at any time published elsewhere without giving credit to this Congress for such article or publication.

11. The Congress reserves the right to reject any paper or other contribution that may be offered to it.

12. Rejections by Sectional Committees will not be final; their decisions will be reviewed by the Committee on Papers and Publications, but rejection by that Committee will be final.

13. Authors of finally rejected contributions will be notified in writing of such rejection immediately after it has been made, and, as far as the Congress is concerned, such final rejection is strictly secret and confidential.

14. Any paper which is of a pronounced polemical, advertising or personal character may be thereby disqualified and for that reason alone rejected, regardless of whatever merit the paper may otherwise possess.

15. The Congress reserves the right to print the full paper only, or the abstract only, or the title only, in each case with the author's name and post-office address.

16. The Congress will not publish a list of rejected papers nor state what papers have been rejected; directly after the closing of the Congress all records relating to rejected papers and like contributions will be destroyed; any and all proceedings as to rejected papers or like contributions, so far as the Congress is concerned, will be strictly secret and confidential.

17. Authors are requested to state on the papers themselves their preferences for the sections in which they wish them to be read; the Congress will respect that request wherever practicable, but reserves the right to assign the paper to any other section that may be deemed more appropriate, and such disposition is final.

18. Authors will *not* receive printer's proofs of their papers or abstracts; nor will their papers or abstracts be revised after receipt by the American committees, except when express authority so to do is given in writing by the author and time and opportunity offer for such revision; authors of manuscripts so revised waive any and all right to complain of or rectify any such revision. (See paragraph 6.)

19. The time consumed in reading or presenting the substance of any paper by an author or his representative at a meeting of a section must not exceed ten (10) minutes; papers or presentations requiring more time than that must be suitably condensed so as to fulfill that requirement.

20. Anyone reading a paper of another's authorship must be fully equipped and prepared to defend the paper in discussion, and no one else will be permitted to read such a paper; an offer to read another's paper is an implied statement to the effect of such preparation.

21. In the absence of an author or his properly equipped representative the paper will be read by title only, and if there be any discussion it must be based upon the paper as printed, because neither the paper itself nor its abstract will be read; exceptions to this rule can be made only under regulations that may be adopted by each sectional executive committee.

22. Discussions of a pronounced polemical, advertising or personal character may be ruled out by the chair on that ground alone and not permitted to appear in the printed record; the ruling of the chair in such matters is final and is not subject to revision or appeal.

23. Participants in discussion will be given an opportunity of editing the manuscript reports of their remarks, but printer's proofs will not necessarily be submitted to them, although wherever practicable they will be so supplied.

The Reaction Between Hydrogen and Carbon Monoxide and the Effect of Steam on Carbon Monoxide at Elevated Temperatures.

In an interesting paper by Mr. ARMAND GAUTIER in the *Comptes Rendus*, vol. 150 (1910), page 1564, the author investigates the question of the possible reduction of CO by hydrogen, with formation of water.

Pure CO, mixed with three times its volume of hydrogen, was heated in a porcelain tube and the temperature recorded

by a Le Chatelier pyrometer. Many tests were made at temperatures from 300° C. to 1250° C.; the mixed and dried gases were circulating with a velocity of 450 cc. to 475 cc. per hour. The water formed was absorbed in a tube filled with P_2O_5 , the CO_2 by potassium hydrate.

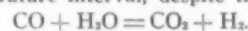
Temperature.	H ₂ O per liter CO.		CO ₂ per liter CO.	
	mg.	cc.	mg.	cc.
400° C.	4.4	5.4	1.7	0.86
600° "	12.4	15.4	11.4	5.80
900° "	17.1	21.2	20.5	10.4
1220° "	22.2	27.5	19.2	9.7
1200° "	17.2	21.3	4.1	2.1

The results show that by heating CO in presence of hydrogen, water is actually formed. The reduction begins approximately at 200° C., before red-heat. The maximum formation of water is between 1100° and 1200° C.

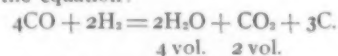
During all the tests the volume of H₂O did not exceed 0.028 of the volume of CO used. At the same time the formation of CO₂ increases till 900°, and drops down when the temperature is increased above this point.



This equation also explains the stability of the water volume in this high-temperature interval, despite the reverse equation.



Between 900° and 1000° the water and CO volumes may follow pretty closely the equation:



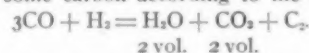
But no trace of carbon was found.

Two remarkable phenomena may be noticed, the decrease in CO₂ between 1200° and 1220° and a similar one during the period of formation of water.

By using a mixture of CO + 3H₂ and a consumption of ½ liter per hour, the author obtained at 900°, under the same conditions, the following results:

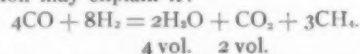
Temperature.	H ₂ O per l CO.		CO ₂ per l Co.	
	mg.	cc.	mg.	cc.
I..... 900°	17.10	21.2	20.5	10.4
II..... 900°	9.3	11.5	21.4	10.8

Test II gave some carbon according to the equation:



Test I showed no gain in carbon. During the latter the water volume was nearly twice the volume of CO₂, and the presence of some methan could be detected.

This equation may explain it:



By heating a mixture of CO and hydrogen at 400° and above only a few thousandths per cent of methan were obtained whose presence had not been detected till then.

Gautier assured himself that during such a process no gas was formed which could be absorbed by bromine or condensed at a low temperature, and that no ethylaldehyde or formic acid was produced, except some traces of formol.

The test was repeated at 1200° with a mixture of CO + 6H₂ (1 liter per hour), and the following result was obtained:



At 1300° and 3 liters per hour:



The velocity of gas passing the tube seems, therefore, to prevent the destruction and prevents the formation of heavy hydrocarbons.

During this last test 8 per cent of the carbon had been transformed into methan. A perceptible formation of other hydrocarbons did not take place.

The results may be summed up as follows:

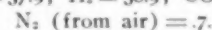
1. The reduction of CO by hydrogen begins approximately at 400°, resulting in the formation of water, carbon and some methan.

2. The mutual reactions are limited and equilibrium is reached at approximately 1000°.

3. The formation of methan is favored by a higher percentage of hydrogen and by a greater velocity to prevent the splitting off of CH₄ at red-heat.

4. Between 450° and 650° traces of formol are formed, but disappear again at 950°.

In regard to the effect of steam on CO at higher temperatures, Gautier made some preliminary tests, using 100 volumes CO with an excess of steam and received the following results:



To prove the probable appearance of formic acid and formaldehyde between 500° and 900° the test was repeated.

The steam condensed after leaving the apparatus contained in 2cc. —0.000005 formol at 550°, at 800° 0.000001 formol, and no formic acid could be found in 2.5 cc. condensed steam.

As to the practical application of the results of Gautier to our metallurgical processes, the reviewer is of the opinion that they may perhaps help to explain the question of formation of hydrogen and methan in blast-furnace gas, which problem has not yet been solved satisfactorily or completely.

They may also be of use in explaining the change in analysis of producer-gas in a long gas main and in the checkers of regenerators. Especially the marked influence of steam on CO seems worth consideration, since a considerable amount of steam is found in all producer-gases.

A similar case is known to the reviewer, where cold coke-oven gas was mixed with hot producer gas in the main of a heating furnace. A report on this is reserved for later.

Sharpening of Tools by Electrolytic Etching*

BY ERICH SCHNECKENBERG.

The attack of an acid on a metal is increased if an electric current is passed simultaneously from the metal into the acid. The current at once carries away all the metallic particles dissolved by the acid, so that the metallic object always presents a clean, fresh surface for the attack of fresh acid.

The current for this purpose may be produced in either of the following ways: 1. A piece of gas retort carbon is placed in the same acid as the metallic object, and the carbon and the metal are connected by a wire outside of the acid, so that the whole forms a short-circuited galvanic cell, in which the current flows from the metal through the acid to the carbon. 2. The metallic object may be connected with the positive pole and another electrode in the acid with the negative pole of some external source of direct current.

In electrolytic etching the attack of the acid upon the tool can be centered and localized on any desired part of the tool, in any direction and strength. All that is needed for this purpose is to form the negative electrode or cathode in a suitable manner and put it in the right position. For in electrolytic etching the quantity of metal etched and removed at any spot is dependent entirely (aside from the time of etching and the concentration of the etching liquid) on the current density at that spot in amperes per square centimeter, and this current density depends on the resistance of the liquid along the various current lines (or tubes of lines) between the positive object and the negative electrode.

For tubes of lines of current of the same cross section this resistance is determined by their length and by the geometrical and physical characteristics of the two opposite surface areas of the metallic object and the negative electrode. These conditions can always be arranged in a suitable manner on the basis of some experiments.

It is possible, for instance, to devise suitable arrangements for sharpening any sort of tool, precision millers, etc. The material for the negative electrode, as well as the acid, must

*Translated from *Werkstattstechnik*, August, 1911.

be chosen accordingly, so as to prevent too rapid destruction of these devices. First of all, I will show how this work should not be carried on.

Several years ago Barthel, in New York, tried the following experiment: He placed some tools, first freed of rust and grease, into a mixture composed of 100 parts of water, 3 parts of 66 per cent sulphuric acid and 6 parts of 44 per cent nitric acid. In the case of tools with two flat surfaces he placed a flat carbon plate opposite to each side surface of the tool. Where round files, millers and drills were to be sharpened he used a cylindrical, hollow carbon tube. When he connected carbon and tool by a wire outside of the acid an electric current passed from the tool through the acid to the carbon.

The etching was, however, different at different spots of the objects, according to the distance and relative position of metal and carbon. Further the carbon also was attacked by the acid; the acid became black in color and the tool was covered with a layer of slime. The tool had, therefore, to be removed after a short time and cleaned, then again placed back and adjusted in position, and etching was resumed, repeating the entire process until the tool had been etched to sufficient sharpness. Then it was rinsed in water, cleaned with a brush, laid in lime water in order to neutralize any possible acid residue and dried with sawdust.

In this form the process was too complicated for workshop use. Moreover, the strength of the current always differed, depending on the surface characteristics, size and number of tools dipped and of the carbon pieces. Since the necessary time of etching depends on the current strength the sharpness of the tool attained by the process can only be estimated from the time consumed, if the current strength is always uniform. The strength of the current could, of course, have been brought to a standard value by connecting several resistances in the exterior circuit, and cutting out some of them when the emf between tool and carbon decreased. But this would have meant that since at the beginning of each etching period the emf was greatest, decreasing then rapidly, a large portion of the emf was to be consumed in the resistance. This, however, had to be avoided since the energy in this cell was anyway rather low.

The introduction of current from an external source (by method two) offers a much simpler and safer process. A storage battery giving 2 to 3 volts is suitable. With this method it is, of course, not necessary to use the most powerful electronegative element, carbon, in order to obtain a sufficiently high emf, but any suitable inert material can be chosen, without any consideration of its position in the contact series. The result is a good, proper etching process for shopwork use.

Grathwol, in Herisau, for example, succeeded in sharpening very dull files within one hour in this way. A patent was allowed him for a file-etching process in which the negative electrode, the cathode, is made of lead, and the electrolyte is sulphuric acid of about 1.18 specific gravity.

Since the quantity of metal etched away at any one place increases with the etching time, it is possible to subject an article to an etching process varying in strength at different places by withdrawing the object gradually from the bath.

Hall and Thornton, in Birmingham, sharpen metal tubes and rods to a point by connecting them with the positive pole of a battery, dipping the end to be pointed into the acid solution connected to the negative pole, and then slowly and gradually withdrawing it from the bath. In this way the rods become thinner and thinner the nearer to the end. This process can be applied to the sharpening of needles, so that thousands of these can be sharpened at one time. All that is needed is a simple trough, and instead of raising the needles gradually from the trough the level of the solution in the trough may be gradually lowered. Another advantage of this method is that the solution in which the metal is dissolved may afterwards be used for electroplating purposes.

Another modification of tool sharpening by electrolytic etching is possible if the tool is brought in contact with the electrolytic acid solution at those places only which are to be etched off, instead of placing the whole tool into the acid solution. This is accomplished in the following way: A mold is made of porous calcined clay corresponding to the desired form of the metal to be etched and is saturated with the etching solution and connected with the negative pole of a battery, and pressed with a slight pressure against the tool, which is connected to the positive pole. After a short time the mold electrode is removed, the tool brushed free of the solution, the mold electrode again dipped into a fresh acid solution, and again pressed against the tool. This method was used successfully by a large electrolytic engraving firm in Leipzig to etch reliefs into metal.

To what accuracy and precision this electrolytic etching process can be brought is proved by the die process of Burdett in Hartford, who etched whole designs, inscriptions and patterns into knives or other metallic surfaces. The metallic surface is covered with a thin layer of a mixture of 10 liters (2.6 gal.) naphthalene, 0.125 kg. (0.27 lb.) carbon disulphide, 2 kg. (4.4 lb.) powdered resin and 1.5 kg. (3.3 lb.) copper chloride. A thin coat of this is put on, and by means of a die which had first been cleaned with a weak caustic potash solution the design is pressed into the coating. Then the particles pressed out by the die are washed off with water; the entire coating is covered with a weak solution of ammonia or brine, a clean metallic surface laid on top and connected to the negative pole of a battery, while the metallic object itself which is to be etched is connected with the positive pole.

Then the electric current passes between the object and the metallic surface, through the ridges cut into the resin coating, so that after a short time the pattern is etched in sharp, clear, deep lines, after which the surface is washed with soda solution. This process also needs only very simple devices to make it fit for operation on a large scale.

Summary.—Electrolytic etching, in which the tool forms the positive electrode for the etching solution in contact with it, permits rapid etching in a simple way, on a definite spot, in a definite direction and strength, depending on the position of the negative electrode, which determines the current flow. It is much better to use an external battery as source of current, instead of generating a current internally from a reaction between tool, acid and carbon. An example for the practical application of tool sharpening by the electrolytic etching method, the electrolytic file sharpening, needle-point sharpening and relief production as well as die sinking work may be mentioned.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Drawn-Wire Tungsten Filament Lamps.

Your correspondent was one of a party of representatives of the technical press who visited the British-Thomson-Houston Company's works at Rugby in August to inspect that company's lamp works and to be one of the intermediaries whereby the announcement was to be communicated to all concerned that drawn-wire tungsten filament lamps were now available in commercial quantities. Possibly a similar announcement and demonstration has been made on your side by the British-Thomson-Houston Company's parent, the General Electric Company.*

*As is well known to our readers, the new drawn-wire tungsten lamp is the outcome of a long series of researches on processes for producing ductile tungsten, carried out at the Research Laboratory of the General Electric Company, at Schenectady, N. Y., under the direction of Dr. W. R. Whitney. The successful work was the result of the efforts of a group of operators and research men, foremost of whom were Dr. C. G. Fink and Dr. W. D. Coolidge.—Editor.

One saw much that was interesting—the new lamps will stand an enormous amount of over-running. It was distinctly novel after one's close acquaintance in the past with carbon filaments, or more recently with squirted tungsten filaments, to see bobbins of tungsten wire from which requisite lengths are cut off as readily as is double cotton-covered wire for the use of armature winders. This is not the place to discuss small structural improvements which, apart from the filament, distinguish this lamp from its predecessors. Neither, and this is the more disappointing, can your correspondent say anything about processes conducted behind locked doors whereby tungsten in its powdered form is converted into a drawn wire. It is a triumph in science, lying on the borderland between chemistry and metallurgy, of great commercial value and more scientific interest. In course of time information, at first no doubt more or less erroneous, must leak out. Meanwhile one can only speculate as to how this has been effected.

A Month of Industrial Unrest.

Early in August the stress of a political crisis masked the symptoms of the labor situation. Unrest and discontent had long been prevalent in South Wales, there had been strikes in the seafaring trade, and employers of labor had recognized that the present trade expansion was liable to be throttled by labor disputes. Just as the Parliamentary situation was clear of all save the recrimination over the *fait accompli*, the trade of London, Liverpool and Leith was paralyzed by strikes of dockers and carmen. On top of this followed a national railway strike, with extensive rioting in South Wales and destruction of property in the Midlands. Considering all the facts of the suddenness of the calamity, the forces of organized civilization in the shape of constabulary and soldiers proved surprisingly adequate. Concessions of increased pay to dock laborers, increased pay and shorter hours for carters and lightermen (certainly justified in these cases) together with the appointment of a Royal Commission to go into the question of the Railway Conciliation Boards have calmed down the situation. The industrial machine therefore is getting into its stride again.

Short though the trouble has been, the loss has been very serious, and will be reflected in balance sheets and trade returns at a later date. Cotton mills and engineering works have been stopped for raw material, contracts have been lost, and the general dislocation will take a lot of adjustment. Of course, the effects of increased wages will lead to a redistribution of the charges incurred thereby; and we shall, unless the index-numbers of the world's food prices take a downward movement, encounter further demands.

The immediate economic questions sink into insignificance compared with the greater sociological questions now partly unveiled to the gaze of the man in the street. To the individualist it is an unwelcome fact that the power of the organized proletariat, by means of a general strike, to attempt to force say state ownership of mines or railways upon the general community brings the climax to the coming industrial struggle thirty years nearer than was anticipated thirty days ago.

The Formation of Sludge in Oil-Cooled Transformers.

An article in the *Times* of the 9th of Aug., by an engineering correspondent, discusses the general aspects of the problem presented to users of oil-cooled transformers, and to the manufacturers of oil for insulating purposes by the deposition of sludge, particularly on the windings and cores. The author says:

"Of the hypotheses advanced to account for the trouble one ascribes most of it to the absorption of vegetable acids from varnishes nominally, but not actually, oil resisting, the balance being attributed to the ingress of dust, which is attracted electrostatically to the windings, and to the thickening of the material through chemical action. The other hypothesis is purely a thermal one, and predicates that a fractional distillation of the oil takes place through high temperatures. In investigating

the problem in any case when the deposit has reached such a stage as to call for a renewal of the oil, the difficulty is encountered that very often little or nothing is known of the initial qualities and subsequent history of the oil, while not sufficient is known of the temperature ranges in the transformers themselves. Probably there are cases which the first hypothesis meets, but there are undoubtedly others to which it cannot apply, and for which the thermal hypothesis is necessary in order to afford a complete explanation of the phenomena observed."

He then proceeds to outline the changes produced in transformer oil by baking at temperature from 230° F. to 270° F., details of which can be found in a paper by Digby and Mellis, published in the Proceedings of the Institution of Electrical Engineers, 1910. The results then given were considered to be mainly of academic interest, but, in connection with the thermal theory of formation of sludge, those results, together with further investigation, might be of very practical value.

The time which elapses before any noticeable deposit of sludge occurs varies considerably. The colloidal deposit has been observed after only six months running of a transformer; but generally the accumulation of a serious deposit occupies from three to six years. Observations with thermo-couples have shown that at points where the core-plates are burred the temperatures are from 15° F to 25° F. higher than the warmest portions of the oil remote from such positions; and, of course, the deposition of sludge prevents the normal distribution of heat by convection and gives rise to a largely extended area in which the heat is unduly augmented.

Analyses of sludges have yielded some very interesting information as to their various metallic and organic constituents; and the research is being continued in the hope of elucidating the causes of a trouble the importance of which is now recognized.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the seven months ended the 31st of July gives the value of the imports of iron and steel, including manufactured articles, at £6,501,659 and of exports as £25,373,018, and the respective increases over the value for the corresponding period of 1910 are £1,511,050 and £485,396, but in July, 1911, there was a deficiency of £727,055 as compared with July, 1910. For other metals, including manufactured articles, the imports were of the value £16,044,594 and the exports £6,322,193, and the increases as compared with the value for last year are £2,697,777, and £463,622. Imports of electrical goods, exclusive of machinery and line wires, amounted to £795,065, a decrease of £46,026, while exports were £1,542,133, or a decrease of £420,556. The value of imports of machinery was £3,575,125, with exports £17,952,018, showing increases of £956,676 and £1,199,728, respectively. Importations of new ships were put at £48,203, and exports at £3,220,582, the imports having improved by £40,856, and the exports having decreased by £1,636,275.

The British Association.

At the meeting at Portsmouth on the 31st Aug., Professor H. H. Turner in his presidential address to the Mathematical and Physical Science section on "The Characteristics of the Observational Sciences" declaimed against the dangerous—he might have said pernicious—doctrine that work done before the formulation of a law was in any way inferior; and instanced the valuable results of W. W. Campbell's observations of the line of sight velocities of stars at Lick Observatory, Professor Schuster's analysis of facts bearing on the periodicity of sun-spots, Groombridge's observations of stars, and M. Teisserene's meteorological observations with balloons.

In so doing the professor was only voicing the opinion of the practical scientist against the tendency only too prevalent at present among some possessing high scientific repute to theorize on any and every subject, formulate a "law" from their questionable speculations, and then cast about for apparent

facts which may by hook or crook be distorted into some semblance of evidence in support of their precious "law." Science is not gaining anything by the substitution of such methods for the true and nature-guided work of men like Davy, Faraday and Tyndal; or we should not find our ears assailed at assemblages of "scientific" men with such onological absurdities as the expression, pretty frequently used of late, "atoms of electricity." A certain person of considerable reputation as a scientist several years ago at an annual meeting of one of the leading London scientific societies uttered the exhortation "therefore, gentlemen, let us DREAM," and it is only too apparent that there are many who do not need such advice to cause them to give preference to methods bordering on the metaphysical.

A Fourth Recalescence Point in Steel.

John O. Arnold (Intern. Zeitsch. Metall., 1911, i, pp. 192-203), as a result of comparing cooling curves of steels of varying carbon content with curves of iron containing no carbon, has found that an evolution of heat occurs with steel between the A_2 and the A_1 points. The greatest disengagement of heat is obtained with steels having a carbon content of about 0.45 per cent., which would consist of approximately equal proportions of pearlite and ferrite. This recalescence the author supposes to be due to constitutional segregation of the solid solution with formation of microscopic distinct masses, beginning at A_2 and ending at A_1 .

Cementation of Chromium Steels.

F. Giolitti and F. Carnevali (Atti. R. Accad. Sci. Torino, 1911, 46 pu. 558-568) have experimented with a chromium steel containing 2.33 per cent. Cr, 0.41 per cent. C, 0.15 per cent. Si, and 1.02 per cent. Mn, by cementation in the same way as that adopted for nickel steel (Journ. Chem. Soc., 1911, ii, p. 609). The results of five hours' treatment in ethylene at 950°C ., and at 1050°C ., treatment with carbon monoxide for the same time at the same temperatures, and with various mixtures of carbon monoxide and carbon dioxide at 1000° and 1100° indicate that the differences observed in the action of these cementing media on carbon steels are produced similarly in chromium steel. The presence of chromium tends to augment the maximum concentration of carbon in the zone of cementation in comparison with the maximum in carbon steel similarly cemented.

Determination of Chromium in Chrome Iron Ore.

Otto Nydegger (Zeitsch. Angew. Chem., 1911, 24, pp. 1163-1164) takes 0.5 gramme of the powdered ore with 5 grammes of a fused and powdered mixture of two parts of anhydrous sodic carbonate with one part borax glass, and heats the mixed ore and fusion-mixture in a platinum crucible over a bunsen flame until all the ore is taken up—from an hour to two hours being required. When cool the mass is dissolved out with 300 cc. or 400 cc. of water containing one volume per hundred of sulphuric acid. Any residual unoxidized chromium is converted into chromate with 0.2 gramme of potassic persulphate, which precipitates, as peroxide, any manganese that may be present. Excess of persulphate is decomposed by boiling for half an hour or so, the MnO_2 is filtered off, and the chromium is estimated volumetrically by ferrous ammonium sulphate with a potassium ferricyanide indicator.

Market Prices During August, 1911.

Copper has been fairly quiet throughout the month. Opening at £56.7.6 it rose to £57 per ton on the 14th, since dropped to about £56.5.0, and closes at £56.2.6.

Tin dropped smartly away from its opening price until it reached £189.10.0 on the 14th. Had risen again to £191.0.0 by the 18th; since irregular, and closes at £189.15.0.

Lead has been fairly steady, with rise of 5/- on the 18th. Opened at £14.5.0, closes at £14.10.0.

Haematite shows a sharp rise from £3.1.3 on the 10th to £3.3.0 on the 18th. Since fell again and closed at £3.2.9.

Scotch Pig rose in sympathy with iron generally from £2.12.1 to £2.13.8 by the 18th, closing at £2.13.3.

Cleveland opened at £2.6.10, showed a sharp drop on the 16th to £2.6.3, afterwards rising to £2.7.6, and closing at £2.6.11.

Rubber (Paraffine) has not exhibited any marked changes, was at 4/6½ on the 16th and 4/9 on the 23d; closed at 4/10.

Alum, lump, loose, per ton.....	£5.12.6
Antimony, black, sulphide powder, per ton.....	22. 0.0
Borax, British refined crystal, per ton.....	16. 0.0
Copper Ore, 10 to 25.2, per unit.....	9/6 to 10.0
Copper Sulphate, per ton.....	19. 5.0
Carbolic Acid, liquid 97/99 per cent, per ton.....	0. 1.6
Creosote, ordinary good liquid, per gal.....	0. 0.2½
Camphor, 1 oz. tablets.....	0. 1.9½
Caustic Soda, Ash, 48 per cent, ordinary, per ton..	5.10.0
Litharge, flake, per ton	16. 0.0
Mica, small slab, original cases, per lb.....	0. 2.0
Naptha, solvent, 90 per cent, 160° C, per gal.....	0. 0.8
Petroleum, Russian Spot.....	0. 0.5½
Sal Ammoniac, lump, firsts, delivered, per ton.....	42. 0.0
Sulphate of Ammonia, f.o.b. Liverpool, per ton....	14. 5.0
Sulphur, recovered, per ton.....	5. 0.0
Shellac, Standard T. N., Orange Spots, per cwt....	4. 4.0
Tin Ore, 70 per cent, per ton.....	£120 to 122. 0.0
Hydrochloric Acid, per cwt.....	5.0
Platinum, per oz.....	8.12.6

Differences noted are:

Rise.	
Creosote, per gal.....	£0. 0.0½
Carbolic Acid, per gal.....	0. 0.2
Litharge, per ton.....	0. 5.0
Sulphate of Ammonia, per ton.....	0. 5.0
Tin, per ton.....	0.15.0
Lead, per ton.....	0. 5.0
Haematite	0. 1.4½
Scotch Pig	0. 0.5
Rubber	0. 0.2
Fall.	
Copper Sulphate, per ton.....	£0. 8.0
Naptha, solvent, per gal.....	0. 0.1
Tin Ore	3. 0.0
Copper, per ton.....	0. 5.0

Newcastle Meeting of the British Institute of Metals.

The autumn meeting of the (British) Institute of Metals was held at Newcastle-on-Tyne on Sept. 20, 21 and 22; that is, practically simultaneously with the Toronto meeting of the American Electrochemical Society.

We reserve an account of the excursions, visits and social functions for our next issue, but give herewith concise abstracts of the papers presented.

This young but exceedingly active society must be congratulated on the high intrinsic value of the papers presented at its meetings. The society continues to live up to the high standard set right at its start and it will be seen that the last meeting was no exception.

The headquarters of the Institute, with the office of its very active secretary, Mr. G. Shaw Scott, are at Caxton House, Westminster, S. W. London.

Alloys of Copper and Tin.

A paper by Mr. J. L. HAUGHTON and Prof. T. TURNER dealt with "volume changes in the alloys of copper with tin"

The work of which Mr. Haughton presented an account may be regarded as a continuation of previous researches on the changes in length which occur in a cast bar during and after solidification. To these investigations Picken, Simpson, Murray,

Hague, Ewen and Coe (former research students in the University of Birmingham) have contributed. The method employed has been that of applying an extensometer to one end of a cast bar while the other end was kept fixed. With a single substance, such as a pure metal or a definite compound, the extensometer does not appear to act until immediately after solidification has occurred, and contraction thereafter proceeds regularly. But where internal changes take place due to such causes as the separation of a constituent, or of a eutectic, from the solid matrix, such changes are indicated by changes in length of the bar. The separation of a constituent is generally accompanied by an expansion, while if the interval between the solidus and the liquidus is considerable the expansion of the cast bar is also generally large, and it extends over a considerable interval of time.

The apparatus employed has been modified by the author in order to allow a single operator to observe both pyrometer and extensometer readings at the same instant and to record these results by means of a chronographer on a paper tape. In a few cases tin-copper alloys have been already examined by Ewen and by Wüst, and it is desirable to note that the agreement between the author and these previous workers is generally quite satisfactory.

The paper clearly shows that there are four maxima in the expansion curve and that three of these, with about 10 per cent, 46 per cent and 65 per cent of tin respectively, agree with the "crystallization interval" as determined from the accepted equilibrium diagrams. The maximum observed at 30 per cent of tin is not easy to explain, though it may be pointed out that the equilibrium diagram at this point is exceptionally complicated and that perhaps the "crystallization interval" should extend down to 500° C., in which case the maximum would fall into proper order. Mr. Haughton records the fact that certain of the intermediate alloys of the series are remarkably insoluble in acids and ordinary solvents. Hence the analysis presented unexpected difficulties. These were overcome by the use of bromine and this simple and direct method of solution may be of use to other investigators of the tin-copper series.

Alloys of Aluminium and Zinc.

A paper by Dr. WALTER ROSENHAIN and Mr. S. L. ARCHBUTT, both of the (British) National Physical Laboratory, dealt with "the alloys of aluminium and zinc."

The paper describes a detailed investigation of the constitution of the alloys of aluminium and zinc which was undertaken in connection with an extended research on the light alloys of aluminium—the latter research being carried out by the authors on behalf of the alloys research committee of the Institution of Mechanical Engineers.

The present paper describes the detailed thermal and microscopic study of the alloys. A long series of cooling curves were taken at a very slow rate of cooling, and in many cases the ingots of the alloys were subjected to prolonged annealing before the cooling curves were taken. The microscopic examination of the slowly cooled alloys was supplemented by the study of specimens which had been annealed and quenched at various definite temperatures.

The results are embodied in an equilibrium diagram which differs considerably from the diagram of Shepherd, which has hitherto been accepted; these differences result from the discovery of the existence of the definite compound Al_2Zn_3 which has a stable existence only between 443° C. and 256° C.; in the majority of the alloys this compound is only slowly formed by a reaction between crystals of zinc and an aluminium-rich mother liquor at 443° C. At 256° C., however, the compound again decomposes into practically pure zinc and an aluminium-rich solid solution; this decomposition is, however, of special interest because it produces a finely laminated constituent closely resembling the "pearlite" of carbon steels. The research has been carried out in the metallurgy department of the National Physical Laboratory.

Non-Ferrous Metals in Railway Work.

A paper by Mr. GEORGE HUGHES dealt with "non-ferrous metals in railway work."

In this paper the author has placed on record some of the methods of working and uses pertaining to the non-ferrous metals in locomotive and carriage construction.

The use of copper is described in the shape of plates, stays and tubes, and analyses, tests, specifications, together with life of material and instances of wear and failure, are described.

Copper fireboxes on the Lancashire & Yorkshire Railway have a life of from 10 to 16½ years, an average mileage for the older engines with 140 lb. pressure being 310,000 miles. Engines with 160 lb. pressure run from 235,000 to 272,000 miles. Individual tube plates show a life of 7¾ years.

Workshop methods, such as patching of copper firebox plates, brazing copper steam pipes, melting and mixtures of brass alloys, are dealt with, with the object of eliciting a practical and scientific discussion.

The molding and coring of an injector casting are referred to.

The author states that brass slide valves of the unbalanced type will run 1000 miles with a wear of 0.035 in. on the face. Balanced valves wear 0.0053 in. per 1000 miles, while even better results are showing from a modified form of valve for superheat designed by the author.

A short account of the bearing metals used under the name of white metals is included, together with a note of non-ferrous metals and alloys used in the railway carriage department.

Corrosion of Brass.

A paper of Mr. PAUL T. BRUHL dealt with "the corrosion of brass, with special reference to condenser tubes."

It is urged by the author that so important a subject as the corrosion of brass by sea water, the neglect of which must mean no slight expenditure, should induce steamship companies to keep records bearing on the subject. It has been this lack of data that has hitherto retarded the solution of the corrosion problem. All the companies with whom correspondence was opened possessed "no definite information."

The conclusions arrived at by the author are:

1. That the presence of air or an increase of temperature up to a certain point accelerates corrosion.
2. That iron, nickel and small amounts of lead are injurious; tin up to 1 per cent, large amounts of lead and aluminium are useful in diminishing corrosion.
3. That the inlet pipe and the condenser plates should preferably be made of brass.
4. That the condenser should be protected against stray currents.
5. Protective coatings are not recommended.
6. The importance of "spills" cannot be exaggerated.
7. That the tubes should be flushed with clean water after use.

Hard-Drawn Copper.

A paper by Mr. D. R. PYE discussed "the mechanical properties of hard-drawn copper."

In this paper the lack of any satisfactory definition of standard hard-drawn copper is pointed out. Although the (British) Post Office specification is highly satisfactory for the sizes of wire used in telephone and telegraph construction, it is quite unsuitable for larger sizes. The various definitions adopted from time to time are reviewed, and it is suggested that the one at present adopted by the Engineering Standards Committee, viz.: "The term hard-drawn copper wire shall apply to copper wire which does not elongate more than 4 per cent on a gage length of 10 in. when broken by tension," though not contradictory of the properties of hard-drawn copper, is unsatisfactory since a specification should preferably fix a *minimum* elongation as a safeguard against the wire becoming brittle. An important point is that tensile strength per square inch and elongation at fracture both depend on the diameter of the wire.

Experiments are described confirming a suggestion made by Mr. A. P. Trotter that this tensile strength per square inch diminishes with increase of diameter according to a linear law. It is also shown that the elongation at fracture for similarly manufactured wires depends very much on the diameter, being considerably greater for larger sizes of wire.

In view of these experiments it is suggested that a satisfactory definition of hard-drawn copper wire would fix a minimum tensile strength per square inch given by the formula $T = 30 - 20 D$ and a minimum elongation per cent at fracture given by the formula $e = 5 D$ where D = diameter in inches.

This specification would hold good for all sizes up to 0.5 in. diameter. Finally this suggested definition is compared with the specifications of the American Society for the Testing of Materials, and an appendix is added, giving the results of some recent experiments in America showing the effect of hard-drawing upon the electrical properties of copper.

Electric Conductivity and Constitution.

A paper by Dr. W. M. GUERTLER dealt with "the electric conductivity and constitution of alloys."

The aim of this paper, which is one mainly of theoretical interest, is to call attention to the property of electrical conductivity in its bearing upon the practical determination of the constitution of alloys.

The so-called "equilibrium diagram" for alloys is now almost universally accepted as the basis for any systematic study of the properties of alloys, and hence it is of great importance to be able to construct it accurately, and to have a thorough knowledge of it. The relationship between concentration, temperature and any given property—electrical conductivity, specific volume, magnetism, etc.—is best and most clearly brought out by the employment of a co-ordinate system in those dimensions. Thus taking concentration and temperature as the base, we can erect ordinates from it whose lengths are proportional to the numerical values of the property under consideration, and so obtain a series of intersecting curved surfaces, each of which corresponds to a particular phase-region of the equilibrium diagram.

The converse problem is one of great importance, viz.: that of obtaining the equilibrium diagram (when it is unknown) by projection from these surfaces on to the basal plane.

The best known method, and one which has been hitherto and is still most universally employed for the determination of the equilibrium diagram, is that known by the name of "thermal analysis," in which special account is taken of the variations in the "total heat" of the systems under consideration.

The author discusses at length the limitations of "thermal analysis," and in particular explains why it fails in those cases where, as so often happens in practice, a state of complete stable equilibrium is unattainable within a comparatively short interval of time.

The respective merits of electrical conductivity and of specific resistance measurements are set forth and illustrated by numerous examples from practice.

Solid Solutions.

A paper by Mr. C. A. EDWARDS gave "further notes on the nature of solid solutions."

The main object of this paper is to deal with a few points of interest and difference raised in the discussion on a previous paper by Mr. Edwards on the same subject, and also to stimulate further discussion on a matter which is not purely of scientific interest. The main conclusion in the previous paper was that "solid solutions of two metals or intermetallic compounds are intimate crystalline mixtures and while the crystals are so small that the mass appears quite homogeneous they are, nevertheless, sufficiently large to retain their identity." The principal objection to this view was that it is opposed to the phase rule. A general review of solutions—gaseous, liquid

and solid—is given in the present paper, after which review the author concludes that the above idea does not conflict with the phase rule, and in fact the phase rule was never intended to extend in its application to the internal nature of a constituent.

As regards solutions in general Mr. Edwards holds that gaseous, liquid and solid solutions are members of the same family, the different states of these members being due to differences in their molecular aggregation.

Failure of a Brazed Joint.

A paper by Dr. H. LOUIS discussed "the failure of a brazed joint."

The paper gives a brief account of the investigation of the cause of a failure in the braze of a steam pipe on a steamer, undertaken at the instance of the (British) Board of Trade. The author shows that the failure was due to corrosion following certain well defined lines in the brass, and he traces the cause of these lines to the presence of small quantities of lead and tin in the original brazing spelter. The lead-tin alloy separating out between the crystals of brass formed planes of weakness that gave access to the corroding solutions, and this brought about the gradual corrosion of the entire brazing material.

SYNOPSIS OF CURRENT LITERATURE.

Iron and Steel.

Electric Steel.—It is not often that the results of experimental work carried out on the scale of that described by A. MÜLLER in *Stahl und Eisen*, July 20, finds its way into print. The work was that carried out by a German steel company on a three-ton Girod furnace preparatory to equipping their plant with electric furnaces. That connections to such a furnace can play such an important part in the life of the lining and the power consumption of it is well emphasized in the changes made by the author in question on his furnace. Where one connection was made to the top electrode and another to the bottom electrode it was found that the arc was thrown strongly against the opposite side of the furnace to the conductors. The top conductor was then split into two and led to the electrode on opposite sides, a single conductor being used on the bottom electrode. With this system the arc was blown toward the line. The final solution of the problem was to split both the conductors, bring them up to the furnace many feet underneath and then rise vertically, a pair on each side of the furnace, carrying the conductors leading to the top and bottom electrodes just as far as possible side by side. To increase this length the bottom electrodes were in electrical connection with the shell of the furnace so that connection with them could be made high up on the side of the furnace. This splitting of the conductors keeps the arc central and saves half the lining and all trouble with blowing out of the arc. Even the shell of the furnace was finally made of steel of low permeability. The energy saving in the conductor system and shell as revised amounted to 10 per cent over the first installation. The flat circular arc now carried on the furnace means an even greater saving of electrode, as previously when it was thrown off to one side it caused very uneven consumption and consequent waste of electrode.

The present system of working is to charge the furnace with about three tons of open-hearth metal and put on a current of 4000 to 6000 amperes. A highly oxidizing slag is formed and all the phosphorus and some of the sulphur is removed during this oxidizing period. The bath is then deoxidized under a new slag formed of lime, sand, ferro-silicon and petroleum coke or scrap electrode ends. This removes sulphur down to 0.01 per cent. On an average five charges are worked

in 12 hours, though this is dependent on the amount of impurities to be removed.

With the original form of two conductor connections where the arc was thrown to one side, the silica roof over that side, even with air cooling, lasted only 20 heats. With the four conductor system the roof now lasts 60 to 70 charges. Experiments are not yet completed as to the economy of using other materials in place of silica for the roof.

The side walls were first formed by ramming a mixture of magnesia and tar around a core shaped like the inverted frustum of a pyramid. Such a lining cost \$175. It was later determined that dolomite would most economically replace the magnesia so that all linings are now made of this material. They last 120 charges and cost \$85. The total cost of lining, including repairs to slag line between heats, amounts to 30 cents per ton of steel. The area of the ends of the steel electrodes in the bottom bears the ratio of 0.05 to 80 to that of the whole bottom.

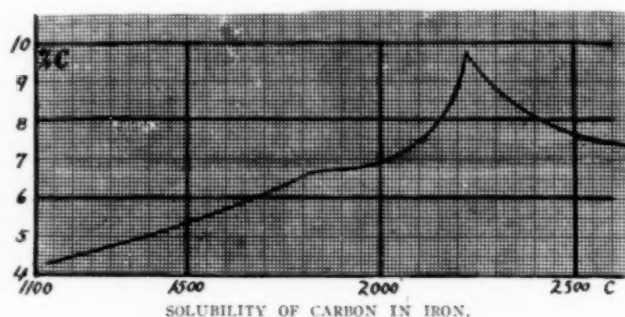
The bottom electrodes are water cooled in the ends projecting outside of the furnace. The dolomite bottom is rammed in between them with an air tamper, such a bottom lasting 1000 heats in a furnace run very intermittently. A calorimetric determination of the heat carried out in the cooling water of these bottom electrodes gave 10.1 kilowatt-hours for the 130-minute run, or about 1.01 per cent of the total electrical energy supplied, or 2.9 kilowatt-hours per ton of steel produced. The cooling water used in the top electrode carried out 36.7 kilowatt-hours, 3.65 per cent of total energy supplied, or 10.5 kilowatt-hours per ton of steel. The paper contains numerous other determinations of the power lost by stoppage, energy requirements for different weights of charges, etc., most of which are given in the form of charts.

The running of this small experimental furnace has thrown much light on the nature of many reactions in the refining of the steel. In the oxidation period carbon is but slowly removed, manganese more rapidly, phosphorus very rapidly and about 20 to 35 per cent of the sulphur. The sulphur appears to be removed in three ways: In the oxidation period by the action of the iron oxide in the slags; in the deoxidation period by solution in the high lime slags and volatilized as silicon sulphide. It has also been noted that oxides, slag emulsions and gases have a much more important bearing on the physical properties of the finished metal than the sulphur and phosphorus present.

Regenerative Furnaces.—A novel change in the construction of regenerative furnaces and one which has proven successful in practice is described by FR. BERHARDT in *Stahl und Eisen*, July 3. The flues carrying the gases from the chambers to the furnace ports are led up separately to the furnace, and distinct from each other. The flues resemble flying buttresses so typical of Gothic architecture in the way they stand out distinct from the furnace. The gas is admitted to the hearth of the furnace in a nearly horizontal stream, while the air is brought down vertically through ports in the roof of the furnace. This construction does away with the narrow division wall between the air and gas ports in the usual style furnace, a source of great trouble at all times. A 17-ton furnace which formerly made on an average of 280 to 300 heats before requiring shutting down for repairs when provided with this new construction ran 630 heats. In a 40-ton furnace the new gas and air flues lasted 1500 to 1800 heats. The ports averaged 280 to 330 charges before requiring any repairs, and after that time the roof was in 50 per cent better condition than that of a similar furnace provided with the old-style ports.

Solubility of Carbon in Iron.—A new determination at high temperatures by RUFF and GOECKE appears in *Metallurgie*, July 22. Iron was heated in a carbon crucible in a carbon resistance furnace. After standing at various temperatures for some time it was dropped into ice water and

the carbon content determined. Temperatures were measured by the Wanner pyrometer. The results are shown in the accompanying curve, the per cent of dissolved carbon forming



the ordinates and the temperature in centigrade degrees the abscissas. The change in direction at 1837° corresponds to the formula Fe_3C , the maximum at 2220° to the formula Fe_2C .

Copper.

Leady Copper Matte.—The treatment of leady copper mattes has always presented a serious problem to the metallurgist. Some new experiments which have led to the introduction of a new process are described by ROBERT HESSE in the June numbers of *Metallurgie*. The author started on the assumption that by smelting a leady matte with oxides of copper, lead and iron the lead could be removed in the metallic state, leaving the copper in the form of an enriched matte, the reactions involved being similar to those in the roast-reaction processes in use in lead and copper metallurgy. Numerous experiments proved the fallacy of this line of work as the only results were the production of a lead-copper alloy, and even that only when the amount of oxide present was very greatly in excess. However, when silica was added to the mixture of matte and oxides, or of roasted matte, black copper containing little lead could be produced and the larger part of the lead and iron slags off as a silicate. Even when an insufficient amount of oxide is present black copper will be produced together with an enriched matte containing but little lead. Low-grade mattes must be concentrated before the process is applicable, the author having great success with Octavi matte containing 40 per cent copper, 24 per cent lead and 11 per cent iron.

As the method does not require resmelting of anything but the lead silicate slag, which can be done in a blast furnace, it represents a saving of 60 per cent of the present cost of treatment as carried out at Altenau. An interesting conclusion drawn from his work is that with electric energy at a cost of \$24 per kw-year the process of fusion can be more cheaply carried out in an electric furnace in Germany than in a reverberatory. The stages of the process recommended for the treatment of the Octavi matte are as follows: Pulverization to 2 mm, roasting to 4 per cent sulphur, fusion with silica. The black copper is refined, the leady slag going to the lead smelter with the silicate slag formed in the first fusion.

Chloridizing Roasting.—The chloridizing roasting of sulphide ores forms the basis of an extended study by VON KOTHNY in *Metallurgie* for July 8. He observed that when pyrite was heated in a stream of indifferent gas sulphur was first evolved at 200° C. and at 700° the pyrite had been transformed over into simple sulphide of iron. When heated in a current of air it started to transform itself to FeSO_4 at 250°, this being the only reaction taking place up to 290°. At higher temperatures it oxidized mainly to Fe_2O_3 , though the reaction took place very slowly at temperatures below 500°. Any FeSO_4 formed was transformed by the oxygen to basic ferric sulphate which at 530° was decomposed into iron oxide and sulphur trioxide.

By the oxidizing roasting of copper sulphide mixed with iron

oxide and sodium chloride practically all the sulphur goes into sulphate and about half the copper is transformed into chloride. Anhydrous cupric chloride mixed with sodium chloride and heated in a current of air to 250° gives off chlorine. The decomposition of copper sulphate by sodium chloride begins at 280°.

In a study of the reactions involved in chloridizing the author finds that the Hargreaves process by which hydrochloric acid is formed plays no part in converting cupric oxide into chloride. Neither does the direct action of chlorine on cuprous sulphide. Ferric chloride converts copper oxide into chloride very rapidly at temperatures around 500-600°. The formation of copper sulphate by roasting pyrite with copper oxide in the presence of sodium chloride plays no part. At temperatures of 300-600° ferric sulphate converts copper oxide slowly into sulphate. The author concludes that the mechanism of the chloridizing of a pyrite cinder containing a small amount of copper and sulphur is as follows: The copper is present largely as sulphide, which by an oxidizing roasting is converted into sulphate and oxide. Sodium chloride acts directly on the sulphate and ferric chloride on the oxide.

To insure chloridizing of such a material it must be finely ground; a large amount of air must be admitted in the oxidizing roasting period and stirring must be resorted to to insure contact with this oxygen; sufficient pyrite must be present to furnish the required amount of ferric chloride and is best added in a weathered form; for 4 per cent copper content at least 7.5 per cent salt must be added; the process should be carried out at temperatures between 500 and 600°.

Gold and Silver

Amalgamation Practise on the Rand.—In our issue for July, 1911, page 322, we gave a review of the proposals of W. R. Dowling regarding amalgamation of blanket ore. These involved the removal of the plates from in front of the stamps and a large reduction of plate area used following tube milling. A further suggestion was that all amalgamation be carried on in a special plate house and that shaking plates be abandoned in favor of stationary plates. The reasons for Mr. Dowling's proposed changes were fully given in his paper.

In the June, 1911, issue of the *Journal of the Chemical, Metallurgical and Mineral Society of South Africa*, Mr. JAMES E. THOMAS contributes to the discussion of Mr. Dowling's paper and gives some data based on conditions before and after adopting Mr. Dowling's suggestions. These are conveniently tabulated in the following tables and present obvious conclusions.

Under the old system of plates in front of mortar boxes and shaking plates following tube mills the following results were obtained:

Average number of stamps running.....	235
Average aperture of mill screen.....	0.0461 in.
Average ratio of water to ore in mill.....	5.75 to 1
Average duty per stamp per 24 hours.....	9.459 tons
Average number of tube mills (22 ft. x 5 ft. 6 in.)..	7
Average amalgamating plate area in use.....	5276 sq. ft.

The plates in front of mortar boxes accounted for 3351 sq. ft. of the total, and the shaking plates for the balance. The latter had a grade of 10 per cent and the pulp flowing over them a consistency of 2 to 1. The extraction by amalgamation was 56.8 per cent of the gold in the ore sent to the mill and by cyaniding 84.67 per cent of the gold in the cyanide pulp. The total extraction on the ore was 93.38 per cent.

During January and February of this year the plates were removed from the stamp mills and the shaking plates in the tube mill circuit were replaced by stationary plates. Comparative results follow:

Average number of stamps running.....	220
Average aperture of mill screen.....	0.0685 in.
Average ratio of water to ore in mill.....	5.17 to 1
Average duty per stamp per 24 hours.....	10.834 tons

Average number of tube mills (22 ft. x 5 ft. 6 in.).. 10

Average amalgamating plate area in use..... 1700 sq. ft.

The plate area consists of three stationary plates per tube mill with a grade of 18 per cent; ratio of liquid to solid in pulp flowing over plates was about 1.1 to 1. The extraction by amalgamation was 55.52 per cent of the gold in the ore, by cyaniding 87.03 per cent of the gold in the cyanide pulp, and the total extraction was 94.23 per cent of the gold in the ore.

Cyaniding Pyritic Concentrate.—The concentration of black sand on amalgamated plates presents a special problem for the recovery of the gold which it contains, usually as particles of metallic gold attached to particles of pyrite. In the July, 1911, issue of the *Journal of the Chemical, Metallurgical and Mineral Society of South Africa*, Mr. R. LINDSAY describes the method of treatment followed at the Geldenhuis Deep, Ltd. The concentrate treated results from the daily scrape from the mills and the barrel residue. The material is ground in barrels for four or five hours and then cleaned in the usual way over stationary copper plates. The flow from the plates is roughly divided in a settling tank into coarse and fine pulp, the former grading about 45 per cent — 200 and the later 98 per cent — 200. The former amounts to about 250 lb. daily, assaying about 30 oz. gold per ton. This is ground in a small tube mill to minus 200-mesh and passed into a batea containing a bath of 250 oz. mercury. The batea in this instance is used in place of an amalgamated plate and serves as a good amalgamator besides recovering some amalgam and mercury which escaped the first copper plate. From the batea the pulp flows to a tank where it is combined with the portion which was already as fine as minus 200-mesh.

The concentrate from the mill barrels is similarly first run over a plate and then divided into coarse and fine portions. The coarse is tube milled, combined with the fine, forming a pulp which is about minus 200-mesh in grade.

The combined fine pulps of pyritic concentrate from the plates and barrels is then ready for cyanide treatment. For this work an agitating and decanting tank designed by Andrew F. Crosse is used. This tank and a description of its use and operation were noted in this journal, February, 1910, page 72. The tank is conical, 7 ft. 6 in. in diameter and 8 ft. 4 in. deep, and has a content of 122 cu. ft. It has a capacity of 1800 lb. dry weight of material and there is no difficulty in obtaining a clear solution for decanting while air agitation is in progress.

In the present instance the practice is to agitate the charge with air for 12 hours before beginning to wash with weak cyanide solution. The gold goes into solution rapidly on account of the fine state of subdivision of the material, a charge assaying 250 dwt. gold per ton being reduced to less than 30 dwt. after one hour's agitation. At the end of 12 hours the undissolved gold in the concentrate is down to 6 dwt. per ton and the cyanide solution has a gold value of 56 dwt. per ton. Washing then proceeds at the rate of ½ ton solution per hour, agitation going on meanwhile. The quantity of solution required for washing the charge of 1800 lb. is six tons, and at the end of this operation the concentrate residue assays 4 dwt. per ton and the solution 0.08 dwt. per ton. This is about the economic limit to which solution and washing can be carried. The cyanide consumption is 2.75 lb. of 120 per cent KCN. Agitation requires 7 cu. ft. free air per minute.

Treatment of Old Slime by Air Agitation.—It is well known that accumulated slime which has become acid by reason of the oxidation of sulphur to sulphuric acid is not readily treated by the cyanide process on account of the heavy consumption of cyanide and the relatively large quantity of gold left in the residues. With a view to determining the best conditions under which such material could be treated, Mr. J. E. R. ADENDORFF made a number of experiments which are recorded in the July, 1911, issue of the *Journal of the Chemical, Metallurgical and Mineral Society of South Africa*. The material treated was accumulated slime containing 22.7 per cent

moisture, 0.024 per cent acidity as H_2SO_4 , 2.31 dwt. gold per ton. The solution used was precipitated slime solution containing 0.017 per cent free KCN, 0.024 per cent total KCN and 0.02 per cent CaO .

Bottle tests were first made and these demonstrated certain facts, viz., the addition of lead acetate was beneficial in raising extraction; the free cyanide was well protected by the lime present; high residues pointed to a lack of oxygen.

Tests were then made on 100-lb. lots in a small Pachuca agitator. The reactions seemed to be most active within the first hour of treatment, during which the alkalinity and percentage of free cyanide dropped rapidly and had to be increased. About 50 per cent of the gold dissolved went into solution in the first hour and the economic limit of treatment seemed to be eight hours. In one test the original slime assayed 2.53 dwt. gold per ton, and at the end of eight hours' agitation the residue contained 20 dwt. undissolved gold per ton. This test seemed to show that success depended on maintaining a high alkalinity during the first three hours of treatment and that lime should be added to the slime before treating with KCN solution. The beneficial effect of lead acetate also was noted in the agitation experiments.

The author makes no mention of preliminary aeration of the slime in alkaline water before the addition of cyanide, which is a condition of success that other experimenters have laid down with reference to the treatment of acid slime for the purpose of oxidizing the reducers present.

Australian Ball Mill Practice.—Pulverizers other than stamps are meeting with success in various parts of the world, and engineers are showing a commendable tendency to investigate the merits of the newer machines. The mills of Kalgoorlie, Australia, have for some time been using ball mills in dry crushing and with considerable success, as is shown by Mr. M. W. von BERNEWITZ in a contribution to the *Mining Magazine* (London) for August, 1911.

It is doubtful if the ball mill is operated anywhere with greater efficiency and less cost than at Kalgoorlie. The ore ground averages 60 per cent silica and is considered hard to grind. It is dry enough for milling, containing from 0.5 per cent to 1 per cent moisture. Anything above this figure retards the crushing action. The best foundation for these mills has been found to be reinforced concrete blocks, as no other structure has given the required freedom from vibration. Two sizes of Krupp mills are used in Australia having the following dimensions:

	No. 5	No. 8
	Inches	Inches
Diameter	80	106
Width	46	54
Outside screen area.....	46 x 26	54 x 24

Approximately the net screening surface in each mill is 70 sq. ft. and 100 sq. ft. respectively. From 24 r.p.m. to 26 r.p.m. is the correct speed for the No. 5, and from 21 r.p.m. to 24 r.p.m. for the No. 8 mill. They are loaded with from one to two tons of 5-in. steel balls, according to the size and hardness of the ore to be ground. There is no advantage in feeding several sizes of balls. The No. 5 mill takes from 18 hp to 23 hp when in average working condition and the No. 8 consumes 50 hp to 60 hp. Where motor driven the ammeters register 60 amp to 65 amp at 550 volts.

The balls are fed either through the manhole or hopper, and ore is added until the balls are well mixed with it. A mill should not be started without ore and the mass of ore and balls should be evenly distributed with no bare balls showing.

The efficiency of the ball mill over some other forms of grinders, notably stamps, lies in the fact that as soon as the ore is ground fine enough it must pass out of the mill. The ore crushed by the balls passes through the perforated grinding plates and is screened by the inside or coarse screen, the oversize being returned to the mill for further grinding. That

part which passes the inside screen is further screened by the outside screen and the oversize returned to the mill.

Mr. von Bernewitz calls attention to the fact that the grinding of a ball mill generally has been supposed to be due to impact, but from observation he is convinced that this is not so and that the grinding is due to rolling and abrasion. The balls are lifted only to about the height of the mill axle and then roll back, rubbing and grinding the ore. This belief is confirmed by a microscopic examination of particles of various sizes taken from a mill. The larger ones, say plus 30-mesh, are shown to be rounded, while the finer grains, say plus 120-mesh, are angular.

The average consumption of balls is about 0.55 lb. per ton, and the grinding plates last about 190 days. A No. 5 ball mill will crush ordinary sulphide ore up to 43 tons daily and the No. 8 up to 100 tons. Dividing this by the horse-power consumed we get 2.1 tons and 1.6 tons per horse-power respectively. In a small plant of 100 tons daily capacity the No. 5 is a useful unit, but in larger plants the No. 8 is preferable. On quartz the capacity of the No. 5 mill is about 36 tons per day. The capacity of the mills is greatest with a feed of 3-in. material. The cost of ball milling is given by the author for several mills in Kalgoorlie. Depending on the ore treated, this varies about 35 cents to 65 cents per ton.

Wet Method for Making Gold Assays in the Field.—

In a paper read before the Institution of Mining and Metallurgy, London, May 31, 1911, Mr. G. M. AUSTIN gave his experience with a method of assaying for gold ores by a wet process suitable for use in the field where the usual facilities are not at hand. In brief the method is as follows: 1 A. T. of pulverized ore is treated with 50 c.c. of a solution containing 2 parts of iodine and 4 parts of potassium iodide in 100 parts of water. The mixture is ground in a mortar, and if necessary more iodine is added. The color of the mixture should remain brown. After 10 minutes' grinding and one hour standing the mixture is filtered and the solution agitated in a bottle with 3 g. of pure mercury. The color will disappear and the gold will be reduced and collected in the mercury.

The mercury is then separated from the solution and treated in a dish with 10 c.c. of nitric acid. The mercury will dissolve and leave the gold in a compact form ready to be annealed and weighed, or fused into a button and measured, or compared with standard beads mounted on a card.

Mr. Austin's experience with this method on a variety of ores gave the following results. Oxidized ores containing only a trace of gold showed this trace quite plainly. Low results were obtained in every case with basic ores containing manganese. Iron pyritic ores gave rather low results and copper pyritic ores containing as high as 34 dwt. gave no gold even after roasting. Telluride ores gave fair results, but the consumption of iodine was high. Ores containing blende and mispickel gave only about 75 per cent of their gold content.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Desulphurizing and Nodulizing Iron Ores.—For the utilization of pyritic cinder and other finely divided iron ores which are not fit for use in making iron, Mr. TOM C. KING, of Marion, Ala., has invented a desulphurizing and nodulizing kiln which reduces the sulphur content to substantially nothing and discharges a sintered and agglomerated product suitable for blast furnaces. As shown in Fig. 1, the device consists of a rotating kiln suitably mounted and driven, having a hood 8 mounted on a carriage 9, with a fuel and supply pipe 11 projecting through the hood. The upper end of the kiln projects into a boiler setting, this end also being in connection with a feed chute 14. The heated gases issuing from the kiln are used to generate steam in the boiler 15. The inventor claims that he has found by practice that a temperature of 1200° Fahr. to

1500° Fahr. is sufficient to drive off the sulphur contained in the material and that a further temperature of 2000° Fahr. is required to sinter and nodulize the ore. The presence of moisture facilitates the operation, but no other binders or fluxes are used. (995,542, June 20, 1911.)

Gold and Silver.

Regeneration of Cyanide Solutions.—The recovery of the cyanogen present in solution in combination with the various metals it may have dissolved during the treatment of an ore is the object of a process patented by Mr. RAYMOND P. WHELOCK, of Searchlight, Nev. Generally speaking the re-

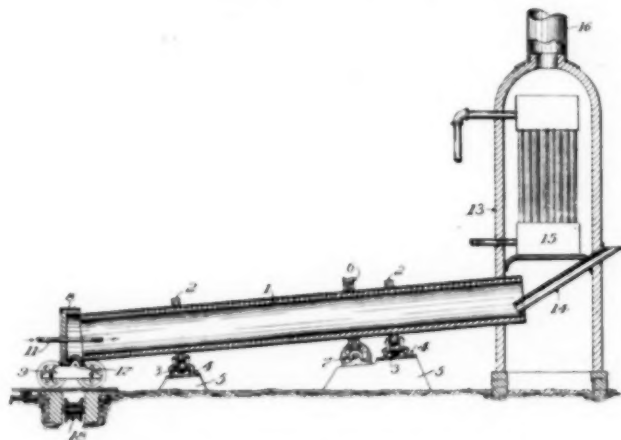


FIG. 1.—DESULPHURIZING AND NODULIZING KILN

sults of the process are first the precipitation and removal of such base metals as copper, lead and zinc which may have been dissolved during the treatment of the ore; second, the precipitation of any gold and silver which has escaped regular precipitation by the regular methods; third, the recovery of the cyanogen which was combined with these metals as soluble compounds.

The first step is to add sufficient sulphuric acid to the spent solution to precipitate as simple cyanides the metals held in solution as double cyanides of the metals and the alkali or alkaline earths present. This will liberate one-half of the cyanogen combined as double cyanides, which will be held in solution as hydrocyanic acid if care is taken not to agitate the solution. Taking copper as a typical case the following equation will show the reaction: $\text{H}_2\text{SO}_4 + \text{K}_2\text{Cu}_2(\text{CN})_4 = \text{K}_2\text{SO}_4 + 2\text{HCN} + 2\text{CuCN}$. The precipitate is removed and the solution is rendered alkaline by the addition of a soluble alkali or alkaline earth, such as calcium hydrate, thus regenerating potassium cyanide as follows: $\text{K}_2\text{SO}_4 + 2\text{HCN} + \text{Ca}(\text{OH})_2 = \text{CaSO}_4 + 2\text{KCN} + 2\text{H}_2\text{O}$. The precipitate resulting from this step is filtered off and the resulting clear solution used again as a solvent for gold and silver.

The precipitate resulting from the addition of sulphuric acid may be further treated with acid and heated to evolve hydrocyanic acid which can be combined with a solution of an alkali for further use. The metals remaining from this treatment can be recovered if their value warrants the operation. (996,179, June 27, 1911.)

Improved Cyanide Process.—Mr. JOHN C. CLANCY, of New York, whose inventions relating to cyanidation have been repeatedly noted in these columns, has patented another procedure based on his discovery that when pulverized ore is treated with a solution of a thiocyanate containing a halogen salt, and electrolyzed, the precious metals only will be dissolved to the exclusion of the baser metals, and that the process may be applied successfully to either rebellious or non-rebellious ores without preliminary roasting.

The following example is given: 2000 lb. of pulverized ore are suspended in a solution of 2 lb. of thiocyanate, 2 lb. of

caustic soda or lime, 1 lb. of potassium iodide, the mixture being in the proportion of two parts of solution to one of ore. This mixture is continuously circulated for a period of eight hours through an electrolyzer having a current density of about 50 amp per square foot of anode surface. About from 90 amp to 100 amp are used per ton of ore. The function of the current is simply to produce nascent oxygen or liberate iodine from the salt. The same solution can be used repeatedly by keeping up the requisite strength of thiocyanate, but no further addition of iodide is required except to compensate for mechanical losses. The reason for using thiocyanate alone or in conjunction with simple cyanide solution in preference to the ordinary alkaline cyanide solution is that, if cyanide solution be used without containing thiocyanate or the addition thereof, the nascent oxygen produced by the current destroys the cyanide by converting it into a cyanate which is not a gold solvent. The following equations represent the reactions which occur: (1) $\text{KCNS} + \text{H}_2\text{O} + 30 = \text{KCN} + \text{H}_2\text{SO}_4$, (2) $\text{KCNS} + \text{KI} + 40 = \text{ICN} + \text{K}_2\text{SO}_4$. (998,749, July 25, 1911.)

Refining Gold and Silver by Distillation in Vacuum.—The present methods of refining gold and silver bullion consist in electrolyzing their solutions. Mr. WILLIAM C. ARSEM, of Schenectady, N. Y., proposes to effect a separation and refinement by distillation at proper temperatures in vacuum. He finds that silver can be completely distilled from an alloy by heating it to a temperature of about 1300° C. in a nearly complete vacuum. Subsequently the gold can be separated from any remaining refractory metals such as platinum by raising the temperature to about 1500° C.

The furnace in which Mr. Arsem carries out his distillation is shown in sectional elevation in Fig. 2. The main envelope

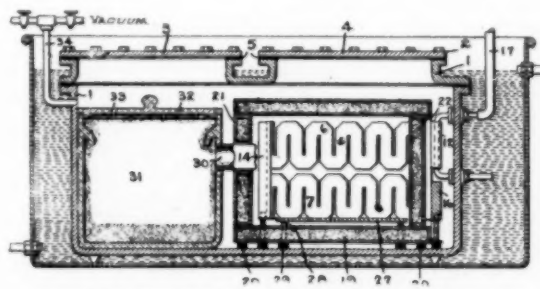
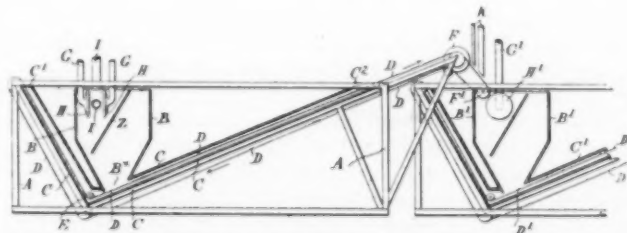


FIG. 2.—SILVER DISTILLATION FURNACE

of the furnace is a rectangular box 1 with a cover 2 bolted to it. The heating element consists of four graphite grids 6, 7, 8, 9, which attain a uniform incandescence when the current passes through them. Current is supplied by way of water-cooled copper tubes passing through the envelope. The heating element is properly insulated by double graphite walls with a



ceived in a container which can be removed from the chamber when the operation of distillation is completed. (908,665, July 25, 1911.)

Separating Solution from Ore Pulp.—A machine designed to afford a simple and inexpensive means of recovering the valuable cyanide solution from treated ore pulp has been patented by HORACE G. NICHOLS, of Ymir, B. C. One of the difficulties of such separation which the machine is designed to overcome is that of causing the settlement of the finely divided pulp held in suspension in the fluid mass. This is accomplished by steadily and continuously withdrawing the solid matter as it settles from the deepest part of the machine by means of an endless moving belt traveling near the bottom of the tank. As shown in Fig. 3, the device consists of a supporting framework *A* in which is mounted a settling tank *B* converging to a bottom opening *Bx*. An endless traveling belt *D* moves along the bottom of the tank, finally emerging with a layer of settled solids and returning to the tank as shown in the drawing. The angle of inclination of the belt is from 20° to 25° with the horizontal. As pulp is fed into the receiving tank *B*, solids settle to the deepest part and are removed continuously by the belt. Valuable solution is withdrawn by means of suction applied through the pipes *G* to the surfaces *H*, the latter being covered with suitable filtering medium. The filtration may be stopped and the accumulated slime blown from the filters by means of a current of any fluid applied in the reverse direction to that of filtration.

The inventor claims that this procedure will remove the solids with a content of 22.5 per cent moisture, and if desired these solids may be similarly treated in a second tank with washing solution for the removal of the contained value. In practice the inventor finds that three of these tanks in succession are sufficient for complete washing. (996,877, July 4, 1911.)

Sand and Slime Classifier.—An improvement in a former machine of the inventor, JOHN V. N. DORR, of Denver, Col., has been devised for the purpose of more completely eliminating moisture or solution from the classified sand. The form of the improved machine is not greatly different from that

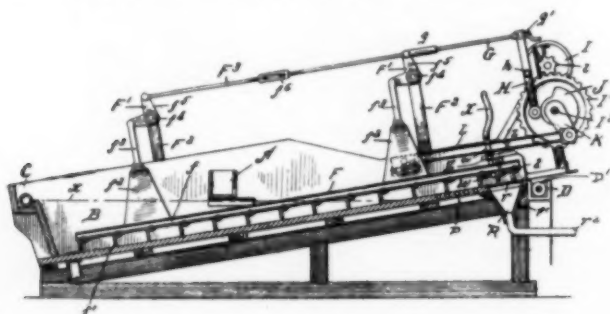


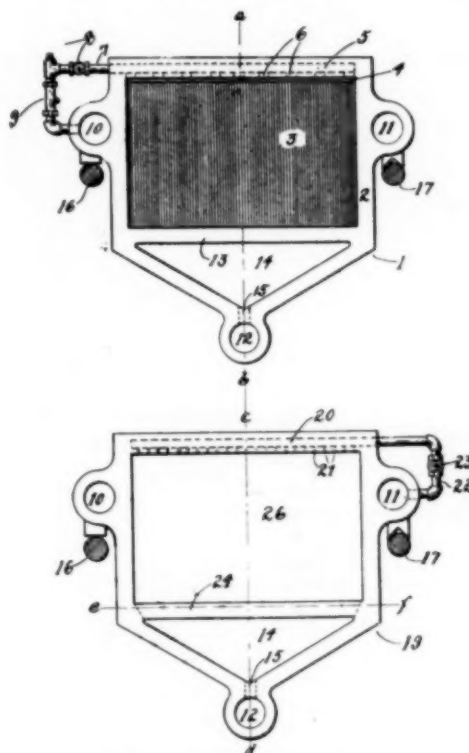
FIG. 4.—SAND AND SLIME CLASSIFIER

of the older, and is shown in Fig. 4. It consists of an inclined settling box *B*, which is fed with the ore pulp through a launder *A*. The liquid level in the box is indicated by the line *x*. Moving within the settling trough is a reciprocating rake which moves settled sand up the inclined bottom of the trough on its forward movement and rises above the sand on its return. The sand is thus continuously removed from the slime which overflows at the launder *C*. Ordinarily in the use of this classifier the sand is removed with about 26 per cent moisture, but it frequently is desirable to deliver drier sand, and it is the purpose of the present improvement to accomplish this end.

Near the point of discharge of the sand, above the level of the liquid in the settling box, will be seen a suction box *R* which is used for the removal of moisture. The suction box is connected with an exhaust pump by pipe *r*². The arrangement and shape of the rakes is such that an even layer of sand is steadily moved onto the surface of the suction box so that the moisture may be withdrawn uniformly. In this way the

sand may be withdrawn with 9 to 10 per cent moisture. If it is desired to wash the sand at the time of discharge this may be accomplished by means of a spray pipe *W*, carried by the rake and arranged to deliver a spray of water on the sand during the upward stroke of the rake. (996,624, July 4, 1911.)

Filter Press.—In METALLURGICAL AND CHEMICAL ENGINEERING for January, 1910, p. 52, there appeared a description of



FIGS. 5 AND 6.—FILTER PRESS

the filter pressing of slime by means of the Sweetland press. The inventor of the press, ERNEST J. SWEETLAND, of Los Angeles, Cal., has recently patented a form of press differing in some features from that described last year. The principle on which the press works can be understood by reference to Figs. 5 and 6, which show respectively a filter frame or drainage unit and a distance frame unit of the press.

The cycle of operations in the press, viz., filling and forming the cake, washing and discharging the same, can be performed without opening or breaking the press. Each of the frames making up the press has three openings at 10, 11 and 12, which register exactly when the frames are locked in the press and form a continuous channel connecting with pipes at the ends of the press. The drainage frame has a rectangular corrugated part 3, connecting with an upper groove 4 and channel 5. The latter is further connected by suitable pipes and fittings 7, 8 and 9, with the opening 10. The part 9 is a glass tube through which the filtrate can be seen as it comes from the unit. The distance frame corresponds in outline with the drainage frame, but is open in the rectangular portion. The opening 11 is connected by suitable pipes and fittings 22, 23, with a channel 20, which has passageways 21 communicating with the rectangular space 26. In both drainage and distance frames the triangular space 14 is open and connected by a passage 15 with the opening 12.

The frames described are placed alternately in a press, and a filter cloth is placed over each side of the drainage unit. The press is filled through the channel 12 at the bottom of the press. The filtered liquid passes through the cloths on the drainage unit, upward through the corrugations 3 to the groove 4 and thence by 5, 6, 7, 8 and 9 to the conduit 10, which has a pipe connection carrying the fluid to the desired place. When

the cake has formed to a sufficient thickness the excess pulp is withdrawn through the conduit 12 and wash water or solution forced in through the same opening. When the pressure is withdrawn from the cakes they are held onto the filter by vacuum applied through conduit 10. In discharging the cakes from the filters, compressed air or water is forced through conduit 10, whereupon the cakes drop into the triangular space 14 and are flushed from the press by means of water jets at one end. At the same time the filter cloths are washed clean by means of jets of water forced through the conduit 11 and openings 21.

A further claim of the inventor is that the triangular space 14 serves as a storage of filtered pulp. In some cases only thin cakes can be formed on the cloths, and then it is more convenient to drop those cakes off into the space 14 and continue filtering without flushing the cakes out of the press. This he claims is possible because the plastic mass of discharged cake does not disintegrate under the action of the inflowing pulp. (996,537, June 27, 1911.)

Copper.

Smelting and Converting.—Mr. GEORGE MITCHELL, of Los Angeles, Cal., proposes to construct a furnace in which reverberatory smelting and converting operations shall be carried on in the same furnace structure, and the heat evolved in the converting process utilized on the smelting hearths. His patent papers show various combinations of hearths and converters, the latter being arranged intermediate the former and adapted to receive matte from hearths by gravity. (997,405, July 11.)

Arsenic.

Recovery of Arsenic from Ores.—An improvement in handling arsenical vapors is claimed in a patent recently granted to IRA F. PECK, of Providence, R. I. The arsenic-bearing material is roasted in a revolving drum and the vapors issuing therefrom are conducted through a series of tanks which contain a thick creamy mixture of lime and water. A vacuum

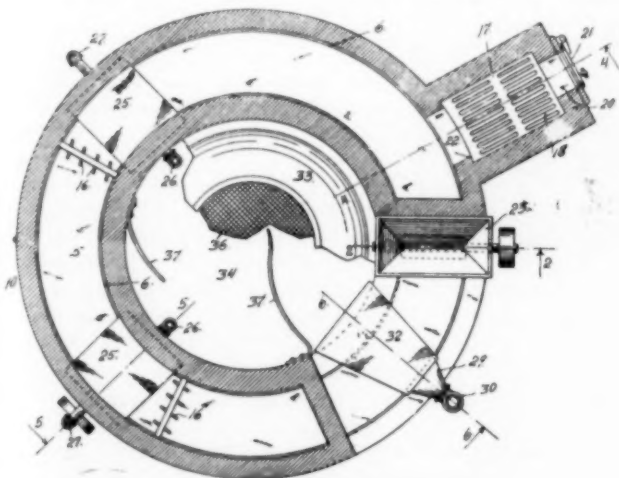


FIG. 7.—MAGNETIC ROASTER

applied at the last tank assists in causing the gas to flow through the lime mixture where the arsenic is absorbed. When the mixture in any tank has taken up as much arsenic as possible it is withdrawn and replaced with a new mixture. The composition of the lime mixture is about 25 lb. lime per 100 gallons of water. The recovered arsenic can be used in the manufacture of insecticides. (996,516, June 27, 1911.)

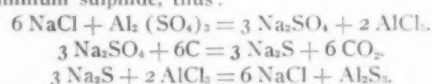
Magnetic Roasting.

Wilfley Roasting Furnace.—In the issue of this journal for April, 1910, page 171, mention was made of a new furnace designed especially for magnetic roasting of ores. A patent on this furnace has recently been issued to ARTHUR R. WILFLEY, of Denver, Col. A plan view of the furnace is shown in

Fig. 7. It consists of a roasting chamber 5 within which is an endless traveling annular hearth. Ore is fed onto the hearth from a hopper 23 having a distributing roll beneath its discharge opening. Heat is supplied to the roasting chamber from a furnace 17 by way of an opening 22. Stationary rabbles 16 are disposed above the hearth at intervals as shown, although if the ore bed is shallow they may not be needed. Receptacles 25 are placed above the hearth as shown, adapted to receive cooling water and thereby regulate the temperature of the hearth at these points. The roasted ore is removed from the hearth by means of a jet of steam delivered through a nozzle 29, this procedure being designed to cool the ore at the time of its removal and thereby prevent further oxidation of the sulphur. The ore thus blown from the hearth passes into the chamber 34, where it is distributed uniformly by the deflectors 37. It is then removed from the chamber in any suitable manner. (996,548, June 27, 1911.)

Aluminum.

The electric reduction of aluminum from its sulphide or a double sulphide of aluminum and another metal is the basis of a patent recently granted to LUCIUS R. KEOGH, of Ottawa, Ontario, Canada. The process consists essentially of causing a reaction between a mass composed of sodium chloride or similar substance and anhydrous aluminium sulphate and carbonaceous matter. Intermediate reactions between these will form aluminium sulphide, thus:



The mass is fused by the application of heat, as by electricity, and the reduction of aluminium will follow. The process can be varied by withholding the carbonaceous material, in which case only aluminium chloride will be formed, which may be distilled away and collected. For the formation of aluminium sulphide and reduction to aluminium the anhydrous sulphate of aluminium is mixed with carbonaceous matter in equivalent proportions, finely ground, mixed with water to form a paste, dried and again finely pulverized. It is then fed into an electric furnace containing a fused bath of the sodium chloride or other selected reagent. The specific gravity of the bath must not be greater than 2.54 in order that the reduced aluminium can sink below it and be tapped from the furnace. The presence of fluorides in the bath contributes to fluidity and salts of lithium decrease the specific gravity of the bath. By regulating the current voltage so as to overcome the resistance of the circuit and just dissociate the aluminium sulphide, no other metal need be deposited, and the aluminium will be obtained free from any appreciable quantity of other metals. (996,094, June 27, 1911.)

Manufacture of Aluminum Nitride.—OTTO KAR SERPEK of Paris, France, has patented a process for producing aluminium nitride from bauxite which he has assigned to Société Generale des Nitrures, of Paris. The general process is as follows: Ground bauxite is calcined, mixed with carbon and heated in an electrical resistance furnace through which is passed a stream of nitrogen generated for the purpose. The apparatus used is designed to afford a methodical heating of the substances with as complete recovery of the heat as possible. Reference to the complete specifications is necessary for a detailed explanation. (996,032, June 20.)

An improved aluminium alloy has been patented in the United States by Conrad H. H. Claessen, of Berlin, Germany. He makes use of magnesium and copper, the former in quantities not exceeding 2 per cent and the latter in quantities not exceeding 5 per cent. He has obtained particularly good results with an alloy consisting substantially of 99.5 to 99.75 per cent Al, 0.25 to 0.5 per cent Mg and 4 per cent Cu. The alloy is claimed to be light, and at the same time stronger and of better quality than either aluminium or the present alloys of that metal. (995,113, June 13, 1911.)

Toronto Meeting of the American Electrochemical Society

The twentieth general meeting of the American Electrochemical Society was held from September 21 to 23 (Thursday to Saturday), at Toronto, Ontario, Canada.

The number of members and guests registered was 125. The attendance was thus somewhat smaller than at most of the recent meetings of the Society, but this was hardly a drawback, as it gave a distinct homelike character to the affair. It was a very congenial gathering and the hospitality of Toronto was unbounded.

There were three professional sessions for the reading and discussion of papers, two on the morning and afternoon of Thursday in the Chemistry and Mining Building of the University of Toronto, and a third at the clubhouse of the Lambton Golf and Country Club on Friday morning. In the whole nineteen papers were presented and discussed, and one more was released and distributed on the train on Saturday morning; printed advance copies were on hand of sixteen papers. In some cases a lively and interesting discussion ensued. A full account of the professional sessions with abstracts of the papers and discussions will be found below.

Smoker.

On the evening of Wednesday a very enjoyable smoker was given to the Society in the West Hall of the Main Building of the University. This will long be remembered for its main features as well as for the quaint arrangement of the whole, with the piper, the torchlight parade, and the song of Auld Lang Syne.

The main entertainment features were a remarkable exhibition of glass-blowing by Prof. E. B. Kenrick and an equally remarkable play by Section Q. Prof. Kenrick showed in the most skillful and entertaining manner the development of three embryos (glass bulbs) into a rooster, a cat, and a human being.

The play by Section Q was entitled "The Electro-Comical Lab," and the playwright actors were Messrs. Lidbury, FitzGerald, Bennie, Saunders, Carveth, Hooker and Dushman. It was an electric-furnace farce, full of bright lines and clever fun.

The rise of the curtain showed an electric furnace room with three assistants getting up things. One has just graduated from a correspondence electric furnace engineering course of three weeks. Instruments are gathered together—an electrode indicator ("going, going, gone") an effishy-ometer, (reading up to 250 per cent efficiency), also samples of electrochemical products. "Priceless" ferrotitanium, for instance (the only ferroalloy Mr. Price does not make). Everything is ready, when the doctor enters with a party of visitors to demonstrate the furnace.

To "graphitize our amorphous ideas" a little text of Brown-ing for Section Q is quoted (back to his studies, fresher than at first, fierce as a dragon, he, soul-hydropic with a sacred thirst, sucked at the flagon).

The demonstration of the furnace begins. Not starting at first it is lubricated with jagdag. The furnace works and the visitors and the doctor talk. A phonograph also talks—an auto-starter, guaranteed to start discussion on any subject at any

time. It is a pity that all the talk cannot be reproduced.

Dr. Zinc O'Chink, the efficiency engineer, is called in, explains water-cooling, sprays the electrode, the electrode breaks and the efficiency rises to 110 per cent. The furnace works now as an induction furnace. Just when the doctor has explained that the furnace is always working very smoothly, there is an explosion. An officer appears and arrests the furnace man—the "pinch effect," also demonstrating "the conductivity of the copper." Mr. Lidbury sings a clever song on making the punishment fit the crime.

The furnace has made graphite. Ferro-alloys are not attempted, since they are all patented several times each, though they are mentioned in the Old Testament. (Didn't Pharaoh's daughter discover Pharaoh Moses in the bull rushes?) So a steel mixture is tried. Fractional distillation of impurities begins, driving off nitrogen and oxides. When the furnace is to be tapped, there is trouble and resort is had to the latest remedy in treatment cases of electrothermic constipation, the squirt effect.

Three ingots of steel are tapped. Owing to the high efficiency of the furnace, the metal is tapped at room temperature, while by watching the stack the visitors see that the furnace is now giving off solid CO₂. The ingots are taken to the rolling mill. The rail is tested, succumbs to the drop test, and is shipped. The demonstration is over. A song on Dr. Heroult is the finale.

It is impossible to do justice to the play in a brief review. But those who had the privilege of seeing it acted, enjoyed it immensely.

Before the smoker an exhibition was given in the basement of the Chemistry Building of the Evans-Stansfield electric furnace for producing titanium steel direct from ore. A feature of the furnace design is the arrangement of the electrodes, several sets being provided for starting and running.

Excursions and Banquet.

Friday was devoted to a trip to the Lambton Golf and Country Club. The professional session of the morning was held at the club house, while after lunch those who wanted to play golf did so on the beautiful grounds. Ideal weather contributed to the enjoyment.

In the evening of Friday a subscription dinner was held at McConkey's restaurant. Prof. Ellis acted as toastmaster with great success. Three toasts were drunk, to the President, to the King, to the Society.

Speeches were made by the president of the Society, Dr. Whitney, by the secretary, Dr. Richards, by the chief engineer of the Hydro-Electric Power Commission of Ontario, Mr. Sothman, by Dr. Baekeland, and by Mr. Cohoe. The two last named speakers called attention to the eminent importance of next year's Congress of Applied Chemistry—the first to be held in this country.

The third day of the meeting was given over to a highly interesting visit to Hamilton, Ontario, and its vicinity. Over forty members and guests participated. The first plant inspected was the large, and rapidly growing, plant of the Canadian Westinghouse Company. The one hour available in this



W. R. WHITNEY, PRESIDENT AMERICAN ELECTRO-CHEMICAL SOCIETY

plant passed all too quickly, among the multitude of interesting and instructive apparatus and operations.

The party then proceeded to the plant of the Steel Company of Canada. Here were two blast furnaces, and numerous basic open-hearth steel furnaces, mixer, etc. The attraction commanding chief attention, however, was the Gayley refrigerating plant for drying the air for the two blast furnaces. These furnaces each require some 20,000 cubic feet of air per minute, which is cooled to 14 deg. F (—10 deg. C.). The moisture content of 6 to 7 grains per cubic foot is thus reduced to 0.5 grain. The output of the two furnaces has been increased from 200 and 300 tons per day to 225 to 350 tons, respectively, while the coke consumption has been considerably reduced, being now 2200 lb. per 2240 lb. of pig iron. The net saving over all in the production of the iron is given as 75 cents per long ton of pig iron. The refrigerating plant contains 48 miles of pipes, and cost \$500,000. The party was divided into groups, and conducted through the refrigerators into the coldest chamber—a unique and long-to-be-remembered experience.

The basic open-hearth steel plant for the inspection of which very little time only was available, contains four furnaces, two of 50 tons each and two of 25 tons. It is fired by natural gas.

The transformer station of the Canadian Hydro-Electric Commission was visited in the afternoon after lunch had been taken at the Waldorf hotel in Hamilton. A spick-and-span new plant is this transformer station, receiving power at 110,000 volts from Niagara and reducing the voltage to 13,200 volts for the local Hamilton service. It is also a switching station for all the territory supplied to the north, up to Toronto. It is equipped to handle 70,000 kilowatts, but only 12,000 kw go through it at present. The line loss from Niagara is given as under 5 per cent. Many obscure phenomena have occurred in the handling of this 110,000-volt current; one being a jump of the current several feet to another crossing line, during a thunderstorm, when either a lightning flash started the current across, or else a condition of unusual atmospheric electric tension superimposed on the 110,000 volts did the mischief.

The party returned to Hamilton, where it broke up and the members went off by train in different directions.

Professional Sessions.

The first professional session was called to order by the president, Dr. W. R. WHITNEY, on Thursday morning. Since the secretary, Dr. J. W. Richards, had not yet returned from Europe, Mr. Carl Hering was elected active secretary for the first sessions. Dr. Richards arrived on Friday morning.

The first paper of the meeting was presented by Mr. Robertson.

Electric Iron Smelting in Sweden.

A paper by Mr. THOMAS DUNCAN ROBERTSON, of the Can-

adian Böving, Co., Ltd., of Toronto, Ont., gave an account of "recent progress in electrical iron smelting in Sweden."

The author referred to Dr. E. Haanel's well-known reports on the work of the three Swedish engineers, Assar Grönwall, Axel Lindblad, and Otto Stalhane in the electro-metallurgy of iron and steel, their first 700-hp electric iron ore reduction furnace at Domnarfvet, and their later 2500-hp furnace at Trollhättan, erected by the Jernkonteret (the association of Swedish Iron Masters).

All these developments have been fully covered in this journal (concerning the design of the Trollhättan furnace and the results obtained at Trollhättan see especially the three articles in our issues of July and September and elsewhere in the present issue).

It may therefore be merely stated here that charcoal is used as fuel and that an important feature of the furnace is the provision made for the circulation of the gases produced by the reduction of the iron ore. The gas is drawn from the cool upper part of the furnace by means of a fan, and blown under the hot roof of the crucible, serving the double purpose of prolonging the life of the roof, and at the same time giving up its heat to, and helping in the reduction of, the charge.

During a large portion of the time the charge has been high in iron and the quantity of slag correspondingly small, so that the iron and slag have been tapped out together, and by means of a small dam the slag has been allowed to run off down a sand trench.

Table I given in Mr. Robertson's paper, is interesting, as it gives a very concise and yet full summary of the working results at Trollhättan.

The thermal efficiency of the furnace was carefully worked out, when it had been running for five months, after which time the lining had worn rather thin, so that radiation losses were probably higher than normal. However, 80 percent (the figure obtained) compares favorably with 82 percent for the efficiency of the Swedish charcoal blast furnace, and with the 70 percent of the English blast furnace.

As the volume of the furnace and gas piping is constant an excess of gas is being continually produced, proportional to the amount of ore reduced. This gas is allowed to burn into the air at present at Trollhättan, but in the newer plants arrangements are to be made for using it for calcining purposes. About 550 cubic meters of gas are produced per ton of pig iron, and each cubic meter will develop 2,100 to 2,500 Calories, as the gas averages about 80 per cent of combustibles.

When calculating the various charges the quantity of charcoal is kept constant at 6½ hectolitres and the amounts of ore and limestone are varied to produce the desired grade of iron, a typical charge at the time when the author was present at

TABLE I.

	PERIOD OF OPERATION					Total.
	1910 *Nov. 15.	1910 Nov. 16 to Feb. 11, 1911	1911 Feb. 11 to Feb. 19.	1911 Feb. 19 to March 19.	1911 March 19 to April 9.	
Per cent. of iron in ore.....	64.92	65.57	65.06	49.50	57.92	61.54
Per cent. of iron in charge.....	59.80	62.10	62.56	42.42	53.06	57.00
Slag per ton of iron (kg.).....	390.	205.	224.	780.	458.	327.
Material charged per hectolitre of charcoal (kg.).....		66.49	71.13	90.31	69.88	70.77
Charcoal per ton of iron (hectolitres).....		24.22	22.47	26.10	26.97	24.79
Charcoal contents:						
Water (kg.).....		69.1	50.8	59.8	40.2	60.9
Gas (kg.).....		41.7	36.9	49.3	43.1	42.9
Ash (kg.).....		11.8	11.0	13.2	17.2	12.8
Coke (kg.).....		293.1	277.6	323.4	325.7	301.4
Total kilogrammes.....		415.7	376.3	445.7	426.2	418.0
Electric Power.						
Time consumed working.....	h. m.	h. m.	h. m.	h. m.	h. m.	h. m.
Time consumed in interruptions.....	7.50	2,009.56	184.32	639.18	506.34	3,348.10
Total time.....	7.50	105.39	4.58	20.57	22.11	153.45
Average load (kw.).....	7.50	2,115.35	189.30	660.15	528.45	3,501.55
Total kw. hours used.....	1,121	1,319	1,694	1,017	1,733	1,344
Kw. hours per ton of iron.....	8,780	2,651,029	312,601	650,480	877,706	4,500,596
Iron per kw. year (tons).....	3,800	2,296	2,149	2,623	2,643	2,391
Electrode consumption:	2.31	3.82	4.08	3.34	3.31	3.66
Gross (kg.).....		13,012	1,578	2,281	2,474	19,345
*Net (kg.).....		6,743	763	1,121	1,285	9,912
Per ton of iron:						
Gross (kg.).....		11.24	10.84	9.19	7.45	10.28
Net (kg.).....		5.83	5.24	4.52	3.87	5.27

*Filling up of furnace. †Net = 51.24 per cent of gross.

the furnace being 266 kilograms of magnetite, 133 kilograms of concentrates, and 11 kilograms mill cinder, the total of these three constituents being 410 kilograms.

To this were added 32 kilograms of limestone and $6\frac{1}{2}$ hectoliters or 100 kilograms of charcoal.

This is dropped into the furnace at one raising of the bell and tappings are made every 18 charges, or about once every six hours.

The conditions governing the grade of iron produced are similar to those in the ordinary blast furnace except that the irregular influence of the air blast is absent. The furnace gives the maximum output when making white iron, as the making of grey iron requires a rather higher temperature and consequently a greater power consumption. By increasing the

stock line and the roof is a space filled with furnace gases which contain about 15 per cent CO_2 , so that the portion of the electrode exposed to them is very gradually oxidized.

The result of this is shown in Fig. 2 where the half-con-

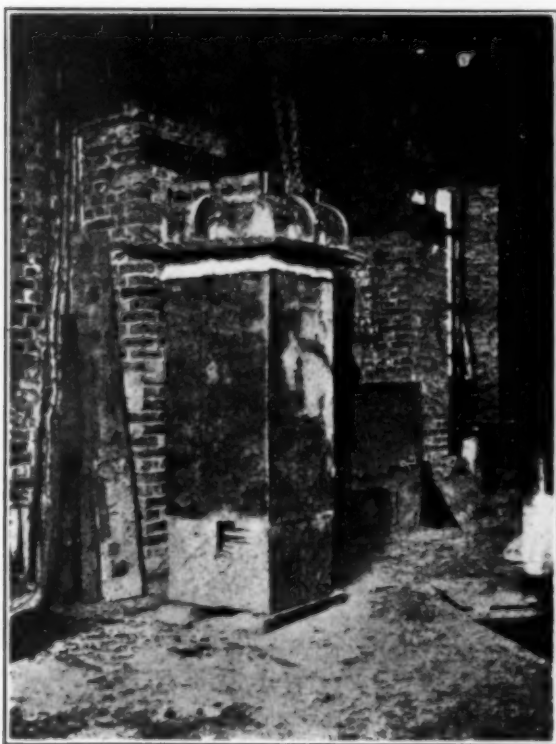


FIG. 1.—NEW ELECTRODE, EQUIPPED WITH IRON SHEATH AND CONTACT FRAME

amount of ore in the charge when the furnace is making white iron, a low-carbon iron with very little silicon is produced, a typical analysis of which is as follows:

	C.	Si.	Mn.	S.	P.
Percentages	2.60	0.10	0.11	0.02	0.01

This iron naturally is full of holes, but instead of these having colored oxidized surfaces, they are silvery white; the absence of oxygen from the furnace atmosphere accounting for the production of this grade of iron free from oxides.

The results of making steel from this iron are given in the second part of the paper.

The tension employed varies greatly and depends mainly on the resistance of the charge. This broadly speaking increases with the amount of slag produced and with the wear of the electrodes, but decreases as the amount of charcoal in the charge becomes greater. The maximum power allowed is 1,900 kw. and to avoid overheating the electrodes the current must not exceed 17,000 amperes, so that the voltage has to be regulated accordingly.

The wearing of the electrodes has given rise to some curious and interesting results. In Fig. 1 is shown a new electrode with its iron sheath and contact frame. When this is placed in the furnace, the bottom end projects into the actual charge and is gradually abraded and consumed. Between the

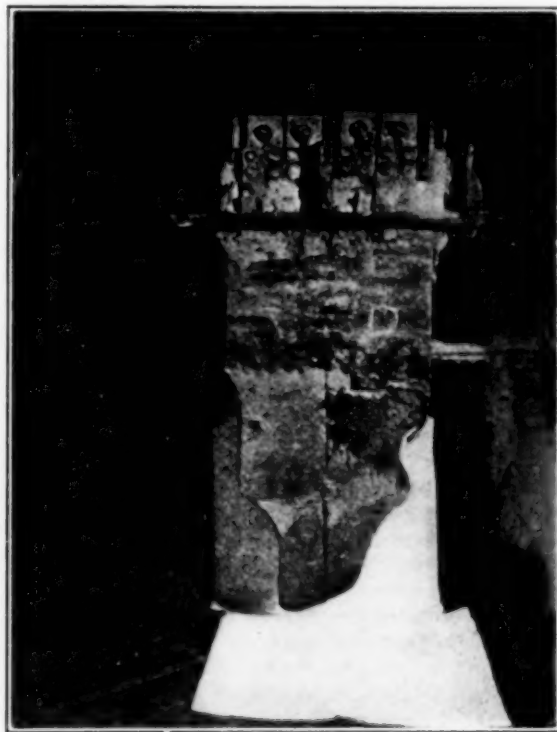


FIG. 2.—HALF-CONSUMED ELECTRODE

sumed electrode was removed for inspection, the line of the

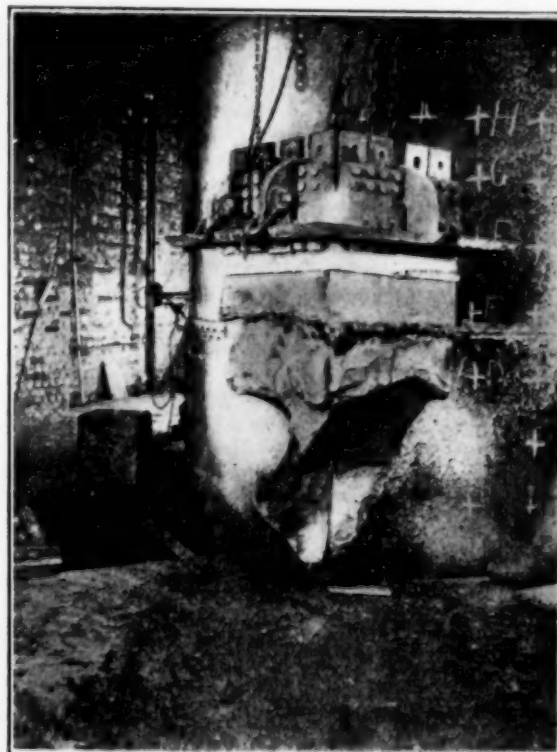


FIG. 3.—STUMP END OF ELECTRODE

surface of the charge being clearly visible in the picture. Fig. 3 shows the stump end of an electrode which had been

lowered as far as possible into the furnace and kept there until unfit for further use. It will be seen that two of the four carbons which compose it have been burnt through to the center.

About three weeks is taken to burn through an electrode above the stock line, so that in practice each electrode is

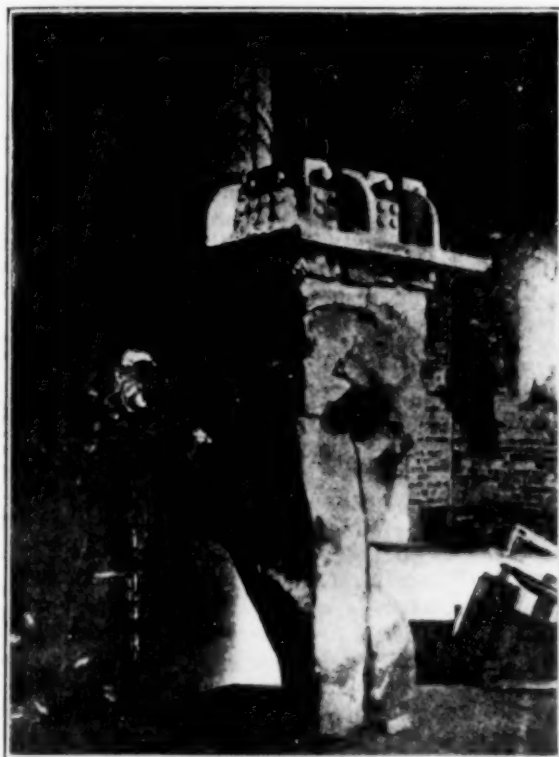


FIG. 4.—EFFECT OF CARBON DIOXIDE ON ELECTRODES

lowered once every two weeks, to ensure regular working without risk of fracture, which would result in a lump of electrode being left in the charge.

With the object of cooling the roof immediately behind the electrodes, gas was blown in beneath the water jackets during a portion of the time. The effect is shown in Fig. 4, the CO_2 in the stream of gas having burnt a hole in the electrode, causing this practice to be discontinued.

A point of interest from a Canadian point of view is the success met with in smelting magnetic concentrates at Trollhättan. The design of shaft in this particular case is not considered suitable for the purpose, being too narrow, but in spite of this, 65 per cent of finely divided concentrates caused no inconvenience in working. The inventors of the furnace are of the opinion that, with a specially designed shaft, charges of all fine concentrates could be smelted.

Canada is rich in deposits of iron sands and lean magnetites, which are easily concentrated, but which are expensive to nodulize or briquette into a form suitable for blast furnace smelting, so that where these are within easy reach of water powers, there seems to be a good field open for electric smelting.

Titaniferous iron ores were found to present no difficulty, although no ores were used with more than 0.8 TiO_2 . Nor does high sulphur content prove objectionable, as there can be little doubt that the electric furnace with its reducing atmosphere and basic lining of the hearth, permitting as it does of the use of a slag very rich in lime, offers the best method of producing sulphur-free pig iron from ores containing that unwelcome element.

The pig iron produced by the furnace was sent to various Swedish iron works for conversion into steel in open-hearth

furnaces. The characteristic feature of electric pig iron is its freedom from oxides; and in consequence electric pig iron of normal silicon content (say, 1 per cent and over) takes a longer time and more ore to convert into steel than ordinary blast-furnace grey iron.

Low-carbon electric pig iron, however, is found to give surprising results, charges made up of 50 per cent of this iron and 50 per cent of scrap producing hot fluid steel with considerable saving of time over ordinary practice.

As was to be expected, the open-hearth furnace managers looked somewhat askance at this iron at first, as they knew the disastrous effect of using low-carbon iron full of holes from the blast furnace; however, after giving it a trial, the workmen asked for more, as they said that the furnace worked better and more rapidly with the new pig iron.

Fortunately for the electric furnace in Sweden, it is more economical to make the white iron, that the steelmakers prefer, than to make high-silicon grey iron. Thus it may be maintained, on the strength of the experience gained, that for the open-hearth process, high-silicon contents are detrimental rather than advantageous, while in the blast furnace pig iron a certain quantity is necessary to neutralize the defects of a reduction process less perfect and ideal than that employed in the electric furnace.

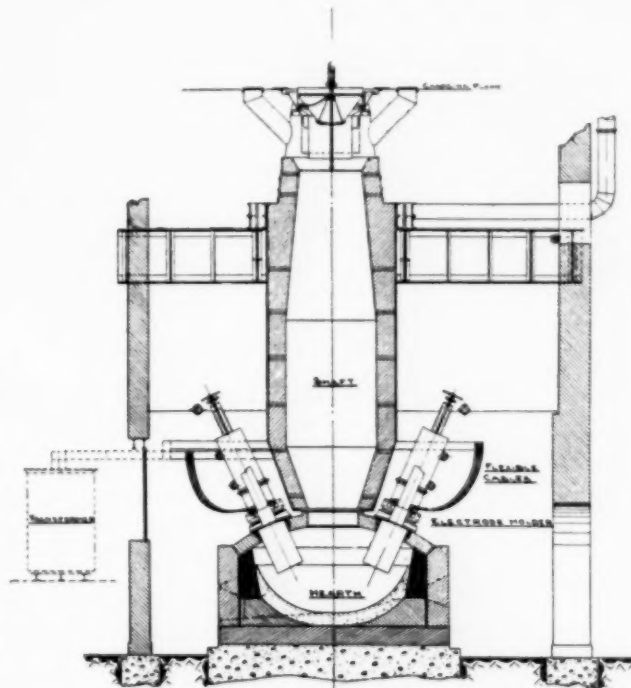


FIG. 5.—DESIGN OF NEW 3000-HP FURNACE

The quality of the steel produced has been thoroughly tested by making it up into various products and comparing it with steel made from Swedish charcoal blast furnace iron. In no case was the new steel inferior, and, from the most recent reports to hand, it was in several cases decidedly superior.

The Trollhättan furnace was shut down at the end of May of this year in order to make certain alterations suggested by the results of its six months' campaign. Two of these deserve mention.

1. Since the furnace was designed, the manufacture of large carbon electrodes, of high conductivity, has made great progress. It is now possible to obtain cylindrical electrodes of 600 mm. diameter, fitted with screw joints. These have recently been installed at Trollhättan, and result in two considerable improvements, the first one being that the loss due to stump ends is done away with, as when an electrode becomes too short, a new one can be screwed on above it. The second

one is that the electrical contact can be made at the point where the electrode enters the roof, in this way saving about 40 kw., which with the old arrangement was lost, due to the resistance of the whole length of the electrode through which the current had to pass. This new arrangement will permit of the furnace taking 3,000 hp. instead of 2,500 hp., and is shown in Fig. 5.

2. The other important alteration is in the gas-circulating arrangement, the new form of which is seen in Fig. 6. The old dust-catcher was not very effective, as only very fine dust is expelled from the furnace. The result was that the dust

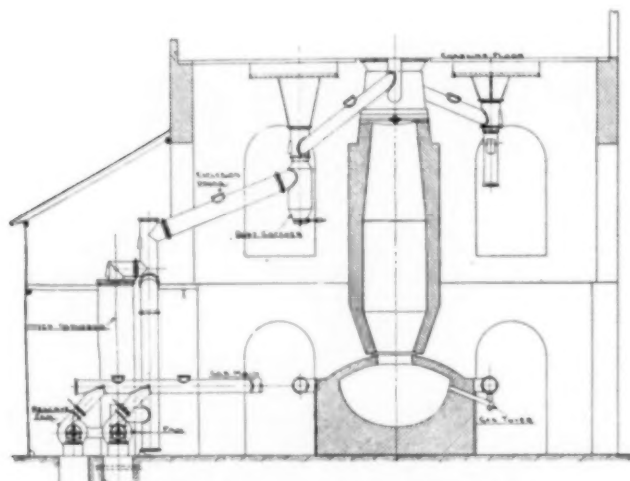


FIG. 6.—NEW GAS-CIRCULATION ARRANGEMENT

collected in the centrifugal fan, and had it not been that this was kept fairly clear by a jet of water, would have choked up the blades in a few hours; as it was the fan had to be stopped and cleaned every week. Now a water-scrubber takes the place of the dust-catcher, and a fan on the ground floor blows the cleaned gas into the furnace, the other fan shown being held in reserve.

Rather more than three metric tons of pig iron are produced by the electric furnace per electrical hp. year, using one ton of charcoal for the reduction of the ore. The blast furnace uses about three times this amount of fuel per ton of iron, so that in Sweden, where, owing to the demand of wood for the pulp industry, the price of charcoal is yearly advancing, there is a great field open for electrical smelting on account of the saving in fuel.

The furnaces in Norway have a somewhat different form of shaft, as coke is the fuel employed for smelting. The volume of the shaft is smaller, but its diameter is greater, than the corresponding shaft of a charcoal furnace. The coke in the charge gives a greater conductivity, so that a lower tension has to be used.

Mr. Robertson summed up as follows:

1. Electric smelting of iron ores is no longer in the experimental stage, but in Sweden is thoroughly established on a commercial scale.

2. The smooth and regular working of the furnace at Trollhättan has been a marked feature, probably less trouble being experienced than would have been the case with a blast furnace doing similar work.

3. The quality of iron produced for steel-making purposes has been shown to be equal and in some cases superior to that of Swedish blast furnace pig iron.

4. The comparative costs of electric and blast furnace smelting depend in general on the cost of suitable fuel and electrical energy, as the electric furnace simply substitutes one electrical hp. year for two metric tons of blast furnace fuel. In Sweden there is no doubt that the electric furnace has come to stay on account of cheaper production.

5. The electric furnace is specially adapted to the smelting of finely divided ores or concentrates.

6. Electric smelting in Sweden has a national value as its adoption will result in the development of water powers which could not in many cases be profitably utilized for any other purpose.

The paper elicited an extended discussion in which Dr. Brady, Dr. Whitney, Mr. Louis Simpson, Mr. Kohn, Mr. Hering, and Mr. Lidbury participated.

Mr. Simpson referred to the troubles due to the dust in the gas and asked whether there was any necessity for using the gas for cooling the roof. This necessity might be questioned since this method is not employed at the Noble electric iron furnace plant in Heroult, California. Another difference from the California practice is that while the electrodes reach deeper into the charge in California, they are free burning in Sweden. Mr. Simpson also questioned whether any reduction of ore was accomplished by the gas in the shaft in view of its relatively low temperature. Finally he stated that English buyers of electric pig iron had found it irregular; while electric pig iron had always been claimed to be of higher quality than ordinary pig iron, it would be most unfortunate if it should turn out to be irregular.

Mr. Robertson replied that the trouble with the dust in the gases existed, even with the new washing arrangement, yet they were continually making progress and improvements and were sure that the problem would soon be solved. As to the use of the furnace gas for cooling the roof, this is considered by the inventors of the Trollhättan furnace to be one of the essential and most important points of their design. While the temperature of the gases in the stack is relatively low, it is in general still sufficiently high for reduction of magnetite. Finally as to the alleged irregularity of the electric pig iron sold in England, Mr. Robertson stated that most of the pig iron made in Trollhättan in the electric furnace was used by the Swedish iron masters themselves, who had supplied the iron ore. None of the Trollhättan electric pig iron was sold in England. Probably the pig iron sold in England to which reference had been made by Mr. Simpson had been made in small electric furnaces, hence the possibility of irregularity. With the new large electric furnace no such difficulty as to irregularity is experienced.

In reply to another question Mr. Robertson stated that at Trollhättan all kinds of electric pig iron are made, low or high in carbon. There is no more difficulty in making high-carbon pig than low-carbon pig in the electric furnace.

In reply to a question of Mr. Hering, how the 80 per cent efficiency of the furnace had been calculated, Mr. Robertson said that he had not the exact method of calculation at hand, but he knew it had been carefully calculated and the three comparative efficiencies of the Swedish electric furnace, the Swedish charcoal blast furnace, and the English blast furnace had been calculated by the same method. Mr. Lidbury remarked that Table I (given above) was so complete as to exact statements of facts that it enabled anyone to calculate any efficiency desired to suit his idiosyncrasies.

Heat Losses of Electric Furnaces.

A paper by Mr. FRANCIS A. J. FITZGERALD, of Niagara Falls, N. Y., is entitled a "note of an unsuccessful furnace experiment." It refers to experiments with various furnaces of the type in which the heat is obtained by downward radiation from electrically heated resistors arranged near the top of the furnace chamber, as described in our vol. VIII, pp. 289 and 317, and vol. IX, p. 259.

Various furnaces of this type have since been built and tested, but the results have been disappointing in one respect. For, while the working of the furnace, so far as its electric properties were concerned, was most satisfactory, the efficiency was very low, owing to excessive heat losses.

The table below gives certain data in reference to five fur-

naces constructed on this principle, the furnace number appearing in the first column. In the second column the nominal electric capacity is given; in the third column the temperature of the hearth at which experiments were made; in the fourth the heat losses in kilowatts, determined by holding the furnace for a long period at a constant temperature and measuring the rate of generation of energy necessary to maintain that temperature constant; the fifth column gives the percentage loss referred to the electric capacity of the furnace. The columns headed "A" and "B" give certain ratios which will be explained later.

Furnace No.	Capacity, Kw.	Temperature, Cent.	Losses, Kw.	Losses, Per Cent.	A.	B.
1	40	1300	18	45	0.20	1.1
2	40	1240	15	38	0.20	1.1
3	150	1250	33	22	0.29	1.0
3	150	1450	42	28	0.29	1.0
4	75	1100	38	51	0.20	2.0
4A	75	1200	35	47	0.22	2.0

Furnace 1 had walls 230 mm. (9 in.) thick. The resistor was "open," that is, it radiated directly on the hearth.

Furnace 2 had walls 345 mm. (13.5 in.) thick, the outer part of the wall being built of hollow bricks. The resistor was "enclosed," that is, there was a septum of refractory material interposed between the resistor and the hearth.

Furnace 3 had walls 345 mm. (13.5 in.) thick, and between these walls and the iron casing of the furnace was a space of 30 mm. (1.2 in.) filled with kieselguhr.

Furnace 4 was of the design described in our vol. IX, p. 259, and had double walls, each 230 mm. (9 in.) thick, separated from one another by an air space 25 mm. (1 in.) wide.

Furnace 4A was the same as Furnace 4, except that part of the walls was covered with a layer of heat insulating bricks 55 mm. (2.2 in.) thick.

Now, since Furnaces 4 and 4A were supposed to be the most improved model of this kind of furnace, the very high heat loss was extremely disappointing. In looking for an explanation of the poor results obtained, the first thing naturally suggested was that the septum between the resistor and the hearth might offer such a high resistance to the flow of heat that the efficiency of the furnace would be low. However, as this hypothesis was formed during the experiment, it was an easy matter to test it by the simple expedient of knocking out the septum. As this produced no appreciable improvement in the working of the furnace, the hypothesis ceased to be useful.

There then seemed to be two other explanations of the phenomenon; either the design of the furnace was radically wrong, or the loss of heat by conduction through the furnace walls was excessive.

The heat energy radiated from the resistor is absorbed by the charge on the hearth of the furnace on the one hand, and by the walls and roof on the other. Now, the heat absorbed by the walls and conducted away to the surroundings of the furnace is lost, and, consequently, other things equal, it is desirable to keep the ratio between the hearth area and the total area receiving heat from the resistor as large as possible. This ratio has been calculated for the various furnaces and is given in the column marked A in the table. It will be observed that the ratio is greatest in the case of Furnace 3, and this is at any rate a partial explanation of this furnace's superiority.

Another important consideration is the ratio between the area of the surface, other than the hearth, receiving the heat radiated from the resistor, and the rate of generation of energy to the resistor. This ratio has been calculated for the various furnaces, and is given in the column marked B. Here it is seen that the ratio for furnace 4 is twice as great as in the case of furnace 3; consequently, other things equal, the heat losses would be twice as great.

It does not follow, however, that, because the ratio B is high, the design of the furnace is faulty, for other considerations may make such a ratio desirable or even necessary. The point that requires close attention is the most perfect heat insulation possible. If the heat insulation is good, then, while

there may be a great difference in temperature between the charge on the hearth of the furnace and the resistor, there will be very little difference between the rest of the surface receiving radiation and the resistor. Here was found the great weakness in furnace 4, for the walls, instead of being good insulators, were relatively good conductors of heat. Observation showed that when a balance had been reached, so that the total energy radiated from the outside of the furnace was equal to the energy generated in the resistor, there was a marked difference between the temperature of the resistor and the interior surface of the furnace. There were no means by which these temperatures could be measured, but by the eye it could be seen that the difference amounted to at least 200° C.

A very clear demonstration that was approximately quantitative of the high heat conductivity of the walls was made by comparing furnace 4A with furnace 4. In furnace 4A the side walls were partly covered with bricks having high heat-insulating properties, about 25 per cent of the total exterior surface of the furnace being protected in this way. From the table it is seen that, working with even a higher temperature than in the case of furnace 4, the losses were considerably reduced. It is unfortunate that the determinations were not made at the same temperature, but it is safe to assume that, if furnace 4 had been tested at a temperature of 1200 deg. C., the losses would have amounted to at least 43 kw. Hence by the improved heat insulation the loss was reduced about 19 per cent.

In building furnace 4, a good grade of firebrick, so far as its refractory properties are concerned, was used. It was recognized that silica brick are better heat insulators than fire brick, as may be seen from Wologdine's determinations,* which show that the heat conductivity of silica brick is 0.0020, and of fire brick 0.0042. On account of its high co-efficient of expansion by heat, silica brick is sometimes objectionable, and it was thought that, with a wall of fire brick 460 mm. (18 in.) thick and an air gap, the heat insulation would be amply sufficient. The air gap, however, was useless, for a reason that has been well expressed by Ray and Kreisinger.** The surface of the inner wall forming the air gap reached a temperature of at least 600 deg. C., so that the very device on which dependence was placed to prevent heat losses was worse than useless.

In the discussion of Mr. FitzGerald's paper Dr. Whitney emphasized that the description of unsuccessful experiments should be encouraged for the valuable lessons which they teach.

Mr. Lidbury read a communicated discussion by Mr. John Thomson, who had been Mr. FitzGerald's co-worker in the design of the furnace. He pointed out two special differences in the design of the later and earlier furnaces, which might possibly have contributed to the difference in the results; he also expressed the opinion that the point made with respect to the air gap was not yet fully proven.

Titanium in Iron and Steel

A paper by Mr. CHARLES V. SLOCUM, of the Titanium Alloy Manufacturing Company, of Pittsburgh, Pa., refers to the work of Mr. Auguste J. Rossi with titanium continued during 30 years or more of his life, until he produced his "titanium alloy" in the electric furnace in 1903. Since that time the manufacture and consumption of titanium in the steel industry has grown with leaps and bounds.

Metallurgists have practically all agreed that the use of a small percentage of an alloy of titanium is of benefit in both iron and steel, but some question has arisen as to the form which this alloy should take. In a recent publication*** the statement has been made that carbon-free metals dissolve more easily than other metals containing carbon, since the latter

*"Electrochem. and Met. Industry," Vol. VII, (1909), page 383.

**"The Flow of Heat Through Furnace Walls," by Walter T. Ray and Henry Kreisinger; Bulletin 8, Bureau of Mines, Washington, 1911.

***H. Goldschmidt in this journal, July issue, p. 348.

contains carbides. The experience of the present author does not bear out this fact, he having found that a carbon-free alloy was not at all adapted to use in the iron and steel industry, and the manufacturers were obliged to make a product containing from 5 per cent to 8 per cent carbon for such use. His early experience with an alloy of iron and titanium free from carbon, but containing 5 per cent to 10 per cent aluminium, resulted in almost complete failure, as the aluminium when present in any appreciable quantity made the steel brittle and its oxide showed great tendency to remain in the steel.

"Titanium alloy" contains 5 per cent to 8 per cent of carbon, mostly in the form of graphite. Analyses of this material made in the laboratory of Dr. C. F. McKenna show the following proportions of the two forms of carbon:

Sample No.	Graphitic Carbon	Combined Carbon
141	9.601	0.147
162	9.179	0.12
291	7.012	0.13
298	6.234	0.118

From these analyses it appears that titanium acts very much like silicon in that it causes a separation of carbon as graphite.

Dr. G. B. Waterhouse recently cited an interesting experiment to bear out this fact: A ladle of iron for making malleable castings was treated with 10 per cent titanium alloy sufficient to equal 0.1 per cent metallic titanium added to the iron. The original iron gave castings showing a white fracture with a slight graphite in the center. The titanium-treated metal showed a white border between $\frac{3}{8}$ and $\frac{1}{2}$ in. (1 to 1.25 cm) deep, and a gray center showing the separation of graphite. Contrary, therefore, to the viewpoint of Dr. Goldschmidt, there need be no fear of the non-solution of the electric-furnace titanium alloy, as it does not contain more than the merest fraction of the unready soluble carbide.

The density of ferro-titanium has an important bearing on its incorporation in the molten bath. Careful determinations made on the electric-furnace product containing both 10 and 15 per cent titaniums, show a density of 6.20 to 6.40. The alloy produced by the aluminothermic process shows a density of between 6.20 and 6.30. The difference between the densities of the two alloys is therefore hardly noticeable.

The alloy is best added to the ladle of steel after tapping from the furnace and before the slag begins to run. For open-hearth steel the supply should be placed convenient to the ladle and shoveled in precisely as so much coal. For soft steel an addition equivalent to 0.03 per cent titanium is sufficient to make the steel more ductile. Larger quantities are of increased benefit, but cannot always be added because of the carbon content and also because of the density imparted to the steel, which increases as the proportion of alloy is increased, and therefore is not always desirable. In high-carbon steels this increased density is desired and more alloy is used. The addition of the alloy permits the use of a considerably higher carbon without increasing the brittleness, and several railroads are now using as much as an equivalent of 0.10 per cent titanium or more in securing tough, durable rails.

In the crucible practice the best results have been obtained by adding the titanium alloy to the pots before they are removed from the furnace, giving more time for washing and deoxidizing before teeming.

Leading authorities seem to agree that titanium achieves its remarkable results through its strong deoxidizing powers, together with its effect of giving the slag formed sufficient fluidity to completely separate it from the metal.

"The presence of titanium oxide lowers the melting point of slags occluded in iron and steel and imparts thereto sufficient fluidity to account for their elimination."²

"Titanium . . . has a stronger affinity for oxygen than have the well-known deoxidizers; . . . it probably gives the

slag such a consistency that it separates more completely from the molten iron."³

"The treatment of all steels with ferro-titanium for the purpose of purifying the metal is strongly recommended, the presence of nitrogen to the extent of 0.02 to 0.035 or 0.045 in certain steels being enough to cause the metal to break asunder, destroying all elongation and reduction of area."⁴

"The great affinity of oxygen and titanium is an absolutely sure means of completely deoxidizing the steel, the advantages of which need no further elaboration. It has been proved that an ingot of steel containing a very small quantity of titanium would show practically no segregation."⁵

The cost of treatment with titanium varies from a minimum of 25 cents to a maximum of \$2 per ton of metal treated. It is the cheapest deoxidizer above the grade of manganese or silicon, and a far greater purifier than any other alloy. The amount to be used is very small, and should be proportioned according to the carbon content of the steel and to the amount of impurities present. To the electrometallurgist it is a triumph worthy of record that Rossi should take an element like titanium, so long considered useless, and make out of it so wonderful a servant in the iron and steel industry.

Mr. Slocum's paper brought forward a lively discussion.

A communicated discussion by Dr. Richard Moldenke, Secretary of the American Foundrymen's Association, was read by Mr. Lidbury. It referred especially to the use of titanium in foundry practice. Dr. Moldenke attributes the success of titanium treatment to its action as a deoxidizer. It removes the oxidation effects which are due not only to remelting under oxidizing conditions (cupola) but can also be traced directly to the blast furnace. The removal of this oxidation, which is small, is important and this is what titanium does.

Because the oxidation is small Dr. Moldenke points out that only a small amount of titanium is needed and useful. If more is added, it makes apparently no difference. This would mean that titanium acts simply as a deoxidizer, not as an alloying element. A practical conclusion from this is that the titanium treatment of metal should be adjusted to the amount of oxidation present.

Mr. Wm. C. Cuntz, general manager of the Goldschmidt Thermit Co., then read a communicated discussion of Dr. Hans Goldschmidt who first replied to Mr. Slocum's claim that a carbon-free ferrotitanium was not adapted to the use in the iron and steel industry and emphasized that this claim was refuted by the undisputed practical results of a large number of prominent metallurgical works which use continually carbon-free ferrotitanium made by the aluminothermic process.

Dr. Goldschmidt does not claim that ferrotitanium containing carbon is unsuitable or useless. But he claims that the carbon-free ferrotitanium, containing aluminium, is a more effective reagent. His first reason is that ferrotitanium which contains little or no aluminium, alloys with steel only with difficulty. When Dr. Goldschmidt had placed on the market in 1897 a 35 to 40 per cent. ferrotitanium containing very little aluminium, an open-hearth steel plant to which he had furnished an alloy of especially high percentage, inquired whether he could not make the product at very low cost, since they were inclined to use this ferrotitanium as an open-hearth furnace lining; so little was it affected by the molten steel. Dr. Goldschmidt could satisfy his steel customers only after he added aluminium to the alloy so that it contained 1 part of aluminium to 3 or 4 parts of titanium.

Dr. Goldschmidt pointed out that the addition of aluminium to the alloy had a threefold purpose. First, it caused the alloy to be embedded and dissolved more easily and intimately in the steel.

Secondly, titanous acid (the product resulting from the titanium reaction in the steel bath) is very difficult to fuse.

²Bradley Stoughton, U. S. Patent Office Proceedings, Ser. No. 463610.

³Henry M. Howe. *Ibid.*

⁴H. le Chatellier. Paper, Congress of Metallurgists, Belgium, 1905.

⁵E. von Maltitz, Stahl und Eisen, No. 41, 1909.

Also alumina (the product resulting from the aluminium reaction in the steel bath) is very difficult to fuse. Hence, if either titanium or aluminium alone are used as deoxidizers, the products of the reaction—titanic acid or alumina, respectively—are liable to remain as suspended particles in the steel, as can be shown by microphotographic methods. But if titanium and aluminium are used together simultaneously, the product of the reaction is titanate of aluminium, which is much more easily fusible than titanic acid or alumina alone. Hence the titanate of aluminium will separate out of the steel bath and pass into the slag much more easily.

Dr. Goldschmidt's third reason for adding aluminium to his ferrotitanium is that when the aluminium serves its purpose of cleaning and deoxidizing it enables the titanium to manifest more strongly its own specific effectiveness.

Dr. Goldschmidt denies that titanium is a stronger deoxidizing agent than aluminium and points out as proof that aluminium reduces titanic acid, but titanium does not reduce alumina. But he agrees that titanium is a good purifier and stands in the first rank as an agent for the production of a dense, non-porous steel. Besides acting as a deoxidizer and for removing nitrogen the titanium itself has some beneficial effect on steel and cast iron.

In the cleansing of the metal bath the great affinity of the added metal for oxygen and its speed of reaction are important, but Dr. Goldschmidt emphasizes that it is undoubtedly the specific properties of the added metal itself which give the result. For example, metallic calcium has a greater affinity for oxygen than aluminium, but in spite of this, though many attempts have been made, calcium is not generally used in steel making. The alloy of calcium and silicon which he has produced by his process as a substitute for aluminium and has put on the market has not shown the results expected and hoped for in spite of the fact that this alloy is decidedly a good deoxidizer. Practical results have not equalled what aluminium alone and what to a far greater extent the aluminium-titanium alloy has accomplished.

Dr. Goldschmidt stated that he had always followed with great interest Mr. Rossi's work in the titanium field. "Mr. Rossi has worked hard to get on the market a high-per cent. ferro-titanium. He had finally to choose a product running 10 to 16% titanium with a high carbon content, in order to make its introduction into steel possible. It is perfectly apparent that a low-percentage alloy, over-ballasted as it is, is not as desirable as a high-percentage alloy. But I do not wish to minimize in any way the services of Mr. Rossi as I appreciate all the difficulties he has overcome in producing the results he has produced."

Dr. Goldschmidt finally referred to the successful use of titanium thermit for the introduction of small quantities of titanium into cast iron and steel, as proof that he has also devoted himself to the metallurgy of titanium for some years and has produced results. Titanium thermit is a thermit which when introduced into iron or steel liberates a fluid alloy of ferrotitanium, while the slag rises to the surface. This has a double effect, an addition of a small amount of titanium to the bath and a violent stirring up with a slight rise of temperature. The quality of cast iron is thereby decidedly improved.

The discussion was continued by Mr. F. A. J. Fitzgerald, who thought that the claimed presence of carbides in ferrotitanium containing carbon was not proven and that the reasons for the claim were weak. He asked: If titanium has really such a tendency to form carbides, why should not carbides be formed if carbon-free ferrotitanium is added to molten steel?

Dr. Waterhouse, of the Lackawanna Steel Company, remarked that the claim as to the carbides had apparently been withdrawn by Dr. Goldschmidt as it was not mentioned in his reply. (To this Mr. Cuntz replied afterwards that Dr. Gold-

schmidt had further investigations on titanium in hand and would give his results on a later occasion.)

While Mr. Slocum recommends that the alloy is best added to the ladle of steel *after* tapping from the furnace and before the slag begins to run, Dr. Waterhouse thinks it is best added *during* tapping.

An important point, according to Dr. Waterhouse, is that time must be allowed for the reactions to go on and for the products of the reactions to separate out.

In his final reply, Mr. Slocum referred to early experiments of Mr. Rossi in the reduction of titaniferous iron ores. He thought that if aluminium is beneficial in the titanium reaction it could be added separately. He finally gave figures on the rapid increase in the production of ferrotitanium at the Niagara Works and its consumption in the steel industry.

Niagara Transforming Stations.

A paper by Mr. A. J. JONES, of Niagara Falls, gives a review of the "transforming stations of Niagara electrochemical and electrometallurgical industries."

Grouped around Niagara Falls are seven hydro-electric generating stations, four of which are on the Canadian side. From these seven stations energy is being used at the rate of 128,000 hp for electrochemical purposes, 56,000 hp for railway service, 36,000 hp for lighting, and 55,000 hp for various industrial services, the total being 275,000 hp or about 5.5 per cent of the available power of the cataract. Of this total amount 146,000 hp is employed locally in industries that have been attracted to Niagara Falls by reason of the generating stations located there, electrochemical processes forming 87 per cent of this amount and 46 per cent of the total amount utilized.

The paper is essentially a catalog of the equipment of the transforming stations of the following companies, with an account of the more salient features of the equipment: International Acheson Graphite Company, Aluminium Company of America, Carborundum Company, Niagara Lead Company, Castner Electrolytic Alkali Company, Norton Company, Union Carbide Company, Niagara Alkali Company, Hooker Electrochemical Company, Niagara Electro-Chemical Company, United States Light and Heating Company, and Electro Metals.

In the discussion which followed, Mr. Fitzgerald pointed out that electrochemical newcomers to Niagara are often astonished at the amount of apparatus they have to put in. The plants at Niagara often have a cramped appearance. Usually not enough space is allowed. This is an important matter.

Dr. Northrup referred to a mercury ammeter, based on the pinch effect and being in use at the Aluminium Company's plant at Massena; it measures 14,000 amp regularly; the capacity being 16,500 amp. The instrument is in continuous operation, no failure having ever occurred.

Production of Molybdenum Steel in the Electric Furnace.

A paper by Messrs. E. T. DITTS and R. G. BOWMAN gave an account of an investigation carried out in the Colorado School of Mines.

The use of a sulphide ore of so active a metal as molybdenum in connection with the manufacture of steel presents a number of problems. Foremost among these are the complete reduction of the molybdenum without serious loss, the diffusion of the molybdenum through the steel to form a homogenous product and the elimination of the sulphur from the steel. The process employed was one based on a reaction described by Mr. F. M. Becket in U. S. Patent 855157, and is believed to be new.

In the production of steel direct from ore in the electric furnace the removal of certain impurities, particularly sulphur, offers difficulties. For the reduction of molybdenite in the presence of molten iron it is necessary to have present some substance which has a greater affinity for sulphur than either molybdenum or iron. The compound formed by the desulphurizing agent and the sulphur must either pass into the

slag or be volatilized as soon as formed. The two metals which seem best adapted for use as desulphurizers, under these conditions, are manganese and silicon.

Manganese in the form of metal, or as the ferroalloy, reacts with sulphides at high temperatures according to the following reaction:



The manganese sulphide forms a slag resembling iron sulphide. If the above reaction takes place in a bath of molten steel or iron the resulting metal is apt to contain small included masses of manganese sulphide. Sulphur in this form has little effect on the properties of steel. This reaction might be applied to the production of alloy steels, such as molybdenum

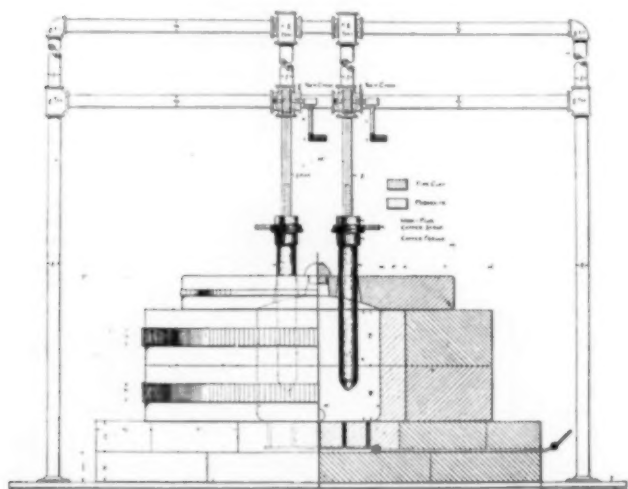


FIG. 1.—LABORATORY FURNACE FOR PRODUCTION OF MOLYBDENUM STEEL

steel, by adding a mixture of molybdenite and ferromanganese to the bath of molten steel just before tapping. This would result in the formation of ferromolybdenum and manganese sulphide; the former alloying with the steel and the latter passing into the slag.

Silicon reacts with sulphides to form silicon sulphide SiS_2 and liberates the metal of the original sulphide. This is the reaction described by Becket.¹ The reaction has been investigated by Fielding² who produced a yellow powder which sublimed at 1500°C and which decomposed water with the formation of H_2S and silicic acid. This compound did not correspond to the formula SiS_2 . Sabatier³ describes a somewhat similar compound and suggests the formula Si_2S_4 . The heat of formation of SiS_2 is given by Sabatier³ as $+40.4$.

This reaction might be applied to the production of molybdenum steel in the same manner as the manganese reaction described above. The silicon, in the form of ferrosilicon, and the molybdenite should be reduced to powder, intimately mixed in the proper proportion to produce the reaction and added to the steel in the furnace before tapping. The mixture might be added in a soft iron tube or in small briquets made up with a binder of sodium silicate. Addition to the ladle during tapping would probably result in raising the sulphur content of the metal on account of the absence of slag, which in the furnace would remove small amounts of sulphur which might tend to pass into the steel.

The experimental work was done in an electric furnace which could be operated either as a Heroult furnace (two arcs in series on the top) or as a Girod furnace (two arcs in parallel on the top and a metallic electrode in the hearth).

The crucible is elliptical in cross section and measures 6 in.

x 9 in. x 0 in. (15 x 23 x 20 cm.) deep. The total volume of the crucible is about 360 cu. in. (5.5 litres), the volume of the smelting zone is about 90 cu. in. (1.4 litres). The walls have a slight bosh to effect a concentration of heat at the base of the crucible. The crucible lining is of burned magnesite obtained by crushing magnesite brick to pass 10 mesh (2.5 mm.) The magnesite was mixed with tar to form a paste and rammed in hot around a central wooden form. The tar was burned out in the heating up of the furnace.

The walls are built up of eight fire-brick sections of special design, luted together with fire-clay and encircled by steel bands. These rest on a foundation made up of two courses of standard fire-brick, with the joints filled with fire-clay.

The cover for the crucible is a solid elliptical fire-brick section 17 in. x 20 in. and $2\frac{1}{2}$ in. (43 x 50 x 53 cm.) thick, encircled by a steel band. The electrodes enter through the slot in the center of the cover. This slot admits of considerable lateral or angular movement of the electrodes. The under side of the cover is slightly concave. An asbestos gasket is placed under the cover

where it rests on the top of the furnace, and small asbestos washers encircle the electrodes where they pass through the cover. The tap-hole is $1\frac{1}{4}$ in. (3 cm.) in diameter, and is well rounded at its point of entrance to the crucible.

The hearth electrode consists of six $\frac{1}{2}$ -in. (1.3 cm.) rods of Swedish iron screwed into a $\frac{3}{8}$ -in. (0.9 cm.) plate of the same material. The plate is embedded in the lining of the bottom of the crucible in such a position that the upper ends of the rods are flush with the inner surface. A copper strap 3-16 in. (0.5 cm.) thick and 1.5 in. (4 cm.) wide, bolted to the plate, extends outside the furnace for connection to the source of current.

The two movable electrodes are supported on a frame above the furnace. The supporting frame is entirely independent of the furnace proper and is therefore unaffected by expansion or contraction of the walls. The electrodes are cylindrical graphite rods 1.25 in. (3 cm.) in diameter and 11 in. (28 cm.) long. Each electrode is attached to a brass rod by means of a collar clamp and cotter pin. The rod carries a rack which meshes with a pinion mounted on the electrode support. The electrode supports are mounted to slide from right to left on the supporting frame. This facilitates adjustment of the electrodes while the surface is in operation and also allows the electrodes to be pushed back out of the way when relining the crucible. The support is so constructed that the electrode may be swung about it as a center and inclined at any angle. Ordinary pipe fittings were used as much as possible in the construction of the frame and supports since these give a light, strong construction, and require no special castings or forgings. An electrode support built entirely of pipe and fittings is illustrated in Fig. 2.

The furnace illustrated in Figs. 2 and 3 differs from that described above in having a crucible of double the depth de-

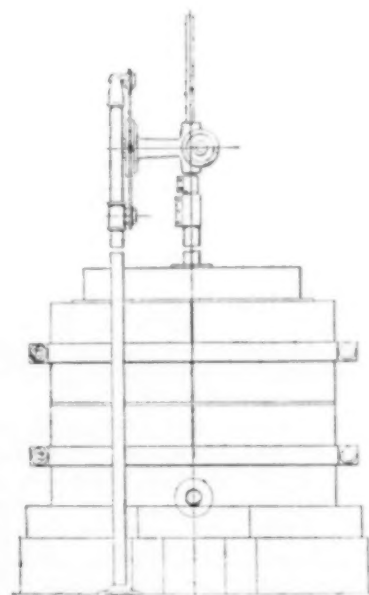


FIG. 2.—MODIFICATION OF FURNACE DESIGN, SHOWING ELECTRODE SUPPORT

¹ Elec. Chem. & Met. Industry, Vol. V, p. 237 (1907.)

² Bul. de Soc. Chem., Paris 2, 38, 153.

³ Comptes Rendus, 90, 819.

scribed, i.e., 16 in. (40 cm.), and a different arrangement of the rods in the bottom of the crucible. This furnace was designed to operate continuously with a column of cold charge resting on the molten material, as in a shaft furnace.

This type of furnace may be operated as a Girod steel furnace by connecting the two movable electrodes in parallel with one side of the circuit and the hearth electrode with the opposite side. The movable electrodes may then be placed parallel vertically, thus forming two arcs, or set to converge to a single central point forming but one arc. To operate the furnace as a Heroult steel furnace, the points of the rods in the hearth electrode are covered with a layer of magnesite, and the two movable electrodes are connected in series with the circuit. The magnesite covering for the hearth electrode need not be of any great thickness, since there is but a slight tendency for the current to cross through the iron plate after the charge becomes molten.

The furnace is charged at the beginning of a run by removing the cover and distributing the charge with a trough or funnel. Additional material may be added through the opening in the cover while the furnace is running.

The small amount of coke in the charge used was found

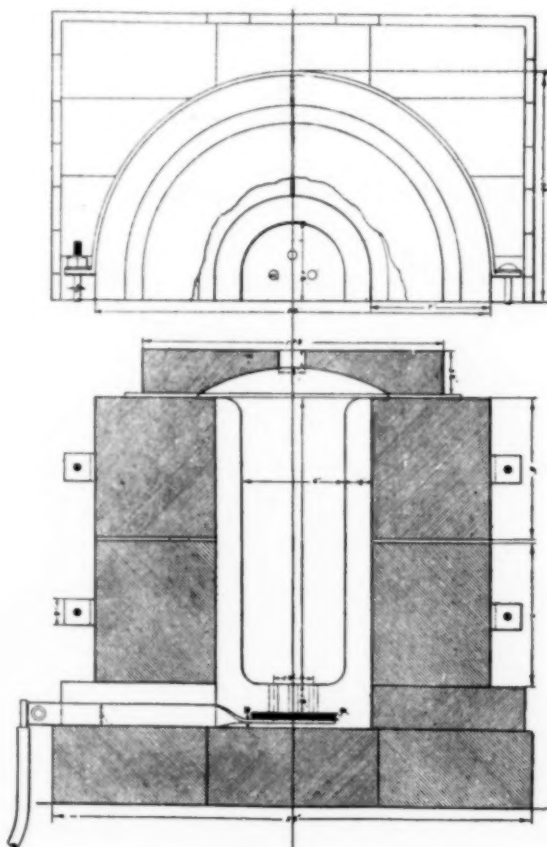


FIG. 3.—MODIFIED FURNACE DESIGN

to be sufficient to make the cold charge slightly conducting, the resistance rapidly decreases with rise of temperature, and the furnace may thus be readily started on a cold storage charge, providing the walls of the crucible are hot. The electrodes are forced down through the charge to within a short distance of the bottom of the crucible at starting. As the charge becomes hot, the electrodes are gradually raised until the charge is reduced and the arcs play between the ends of the electrodes and the surface of the slag. With the current available it was found impossible to maintain arcs below both electrodes simultaneously for any length of time. The best results were obtained by raising and lowering the electrodes alternately at intervals of two or three minutes, which caused

the arcs to alternate from one electrode to the other electrode.

The capacity of the furnace as described is about 1,000 gm of metal per charge, or approximately 2,000 gm. of raw charge.

Single-phase alternating current at 60 cycles, 25 volts, was used, with an average amperage of 250.

The iron ore employed was a good grade of hematite from the mines of the Colorado Fuel and Iron Co. at Sunrise, Wyoming, the analysis being 66.06 per cent Fe., 1.52 Al₂O₃, 4.12 SiO₂, 0.04 P and 0.04 S. The ore was crushed to pass eight mesh (3mm.) and the fines were retained. A quantity of molybdenite from Yerkes Peninsula, Australia, was used, being practically pure MoS₂ with only a trace of impurity in the form of copper sulphide.

Ferrosilicon was used with an analysis of 49.7 per cent Fe, 50 Si, 0.005 S, 0.20 C.

The ferrosilicon and molybdenite were ground to pass thirty mesh (0.85 mm.) and intimately mixed in the proper proportions to bring about the reaction:



The proper amount of this mixture to give the desired molybdenum content in the steel was enclosed in a small paper tube and added to the bath of metal just before reduction was complete.

Analysis of the coke and lime (ordinary builders' lime) are also given. Both were crushed to pass eight mesh (3 mm.) and the fines retained.

Six furnace runs are given in the paper. There was considerable trouble in tapping, as generally the metal would not flow. Only one of the runs is here given.

The charge consisted of 3025 grams iron ore, 660 coke, and 412 lime, the calculated iron content being 2000 grams.

The length of the run was 1 hour, 30 minutes, the mean amperes 200, mean volts 25, power 5 kw.

1,700 grams of metal were obtained. The calculated molybdenum content was 2.5 per cent. The run is described as follows:

Iron turnings were spread over the bottom of the crucible and covered by a layer of coke, lime and silica to form slag and protect the iron. The metal melted in 25 minutes but was kept molten in the furnace for 15 minutes after fusion in order to burn out the tar from new portions of the lining. This metal was tapped cleanly and a charge sufficient for 500 gm. of metal was added. This reduced in 20 minutes. Molybdenite-ferrosilicon mixture sufficient to give a content of 5 per cent. Mo in the metal was added in a paper tube just before reduction was complete. The tap hole was opened but the metal was level with the tap and only slag flowed out. The tap hole was again plugged and a charge sufficient to give 1,000 gm. of metal was charged.

Reduction was rapid and the charge was entirely reduced in 30 minutes. Molybdenite mixture sufficient to give a molybdenum content of 2.5 per cent. in the metal was added just before final reduction of the charge. Since the bottom of the crucible appeared to be sinking constantly, another charge equivalent to 500 gm. of metal was added and reduced in 15 minutes. This gave a total of 2,000 gm. of metal (calculated) in the furnace, with sufficient molybdenum to yield 2.5 per cent. in the total.

The tap hole was opened and slag flowed out freely but no metal was obtained. The current was shut off and the furnace allowed to cool. A solid mass of metal level with the tap hole was found in the bottom of the crucible.

The furnace was taken down and the entire lining and bottom electrode removed. A solid mass of metal weighing 1,700 gm. was obtained. The metal was malleable and tough, and was broken with great difficulty. When broken across the center the fracture was very fine grained and dense at the center of the mass and coarsely crystalline and full of small blow holes at its outer edges.

A section of the central portion of the mass, polished and etched with picric acid showed a network of a bright white

constituent, probably a double carbide of molybdenum and iron. A section in the coarsely crystalline portion of the mass showed a coarsely granular structure.

The analysis of the metal showed 0.62 per cent. C, 0.91 Si, 1.15 Mo, 0.08 P, 0.37 S.

In one of the later runs molybdenite concentrates from Woods flotation process were used. Most of the tests were qualitative rather than quantitative, so that further experiments would be necessary in order to determine the loss and distribution of molybdenum.

Since no analyses were made upon the slag, it is impossible to determine what proportions of the sulphur were removed by the ferrosilicon and the slag.

The following final conclusions are drawn:

(1) Molybdenum steel can be made in the electric furnace by the direct reduction of iron ore and the addition of molybdenum in the form of molybdenite, MoS_2 .

(2) Molybdenum steel of low sulphur content can be produced from molybdenite by the use of ferrosilicon as a desulphurizer.

(3) Molybdenum steel of low sulphur content can be produced from molybdenite in the form of low grade concentrates by the use of ferrosilicon as a desulphurizer.

With regard to the design of a furnace for small scale operations, the experiments seemed to indicate that:

(1) The Girod principle as used was superior to the Heroult.

(2) A tilting furnace would be more effective than a stationary furnace.

(3) The tap hole on a stationary furnace should be made short, with a steep inclination. Ample provision for heating the tap hole should be made.

(4) Tar does not make a satisfactory binding material for crushed magnesite.

In discussing this paper Mr. FitzGerald pointed out there was danger in drawing conclusions from such small-scale experiments. He objected to the use of the names of Heroult and Girod for designating the two respective types of electric furnaces.

Mr. Hansen thought that the use of the two names was useful as a brief name for the two furnace designs, but Mr. FitzGerald objected to it as "electrochemical mythology."

Dr. Whitney liked the paper as coming from two young enthusiastic students. Such work should be encouraged.

Design of a 30-Ton Electric Induction Furnace

A paper by Mr. ALBERT HIORTH, of Kristiania, Norway, describes the design of a 30-ton induction furnace based upon the data and results obtained with the author's original 5-ton induction furnace in Jossingford, Norway.

The two most important things to be determined in the design were the power factor and the energy required to keep the charge at the working temperature.

The power factor of the furnace depends essentially on the ratio of the resistance to the reactance (inductance). The effective resistance consists of three parts, as is indicated by the fact that the power consumption in the furnace consists of three items, two representing the losses, namely, the loss due to the resistance of the primary coils and the hysteresis and eddy-current loss in the iron core; while the third item is the production of useful heat from electrical energy in the (secondary) bath; the secondary resistance may therefore be termed the useful resistance. The useful resistance depends essentially on the conductivity of the iron at the working temperature of the furnace and on the dimensions of the bath.

The author takes from his records figures for the volts, amperes and watts of his 5-ton furnace and calculates the total resistance. From this he deducts the resistance of the primary coils and the resistance corresponding to the losses in the iron core. He thus obtains the equivalent useful resistance of the bath. Figures are given for this useful resistance for different charges in the bath (that is, for different cross-sections of the

molten metal). The resistance should be proportional to the cross-section.

This is indeed approximately fulfilled and from his figures he obtains a specific resistance of the molten iron equal to 1.4 ohms per sq. mm. cross-section and m. length. In handbooks the value given is somewhat higher, and even as high as 1.7. The discrepancy may be due to inaccuracies, which are quite unavoidable, considering the difficult nature of the measurements, or may be due to the fact that in the present case the section of the bath is not absolutely uniform and that, for instance, the central portion of the bath, where the two rings meet, represents a somewhat lower equivalent resistance than the rest. The value 1.40, it must be remembered, includes all the irregularities in the shape of the bath, and also the skin effect, as far as this influences the resistance. It can only be used for figuring the resistance of furnaces of similar shape and disposition of the bath.

The reactance (or inductance) of a given furnace and for a given frequency is independent of the weight of the charge and remains constant all the time.

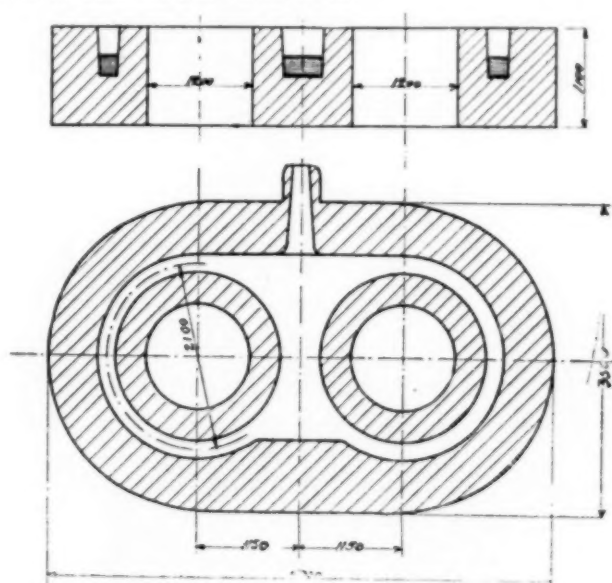


FIG. 1.—DESIGN OF 5-TON FURNACE.

Using the figures thus found for the resistance and inductance of his 5-ton furnace, he figures out theoretically the power-factor of the 5-ton furnace with different charges, and obtains the following table:

With 3 tons	Cos. ϕ	= 0.81.
" 4 "	"	= 0.73.
" 5 "	"	= 0.65.
" 6 "	"	= 0.60.

These figures correspond very well with the actual figures.

With regards to the design of a 30-ton furnace the first condition, of course, is that the space for the charge must be large enough to hold 30 tons, but the actual dimensions and shape of the bath must be such as to give the best possible power-factor.

In order to increase the capacity of the furnace, it is necessary to increase the sectional area of the bath more than the length, as otherwise the diameter of the bath, and consequently the dimensions of the furnace, would become too large. The resistance of the bath will thus be considerably decreased and at the same time the inductance will be increased, owing to the large diameter, which increases the area of the space between the primary and the secondary windings (hence the magnetic leakage).

The power-factor will thus in any case be reduced with the increased capacity, and the author has figured that a 30-ton furnace built on the same line as the 5-ton one, for single-

phase current, with a two-leg magnet and a current supply with 25 alternations, would show a power-factor equal to 0.25, and that the power-factor even with 15 alternations per second would still be only 0.38.

It was therefore decided by the author that the 30-ton furnace should be built on the three-phase principle instead of the single-phase. In this case the weight of the charge per ring is only 10 tons instead of 15 tons with a single-phase furnace and the reduction of the power-factor is not nearly so great.

In order to facilitate the tilting of the furnace, the three rings were disposed in one row.

After having finally chosen the dimensions for his new 30-ton furnace, he calculates the resistance and inductance and finds from these the power-factor which comes out as 0.34, which is still too low.

In order to increase the power-factor, it is necessary to further decrease the inductance by using a lower periodicity. What can be obtained in this way is shown in the following table:

Periodicity	Power-factor
25 alts.	0.34
20 "	0.41
16 "	0.50
10 "	0.67

From this table he selects a periodicity of 16 alternations per second. To decrease the periodicity further still will hardly

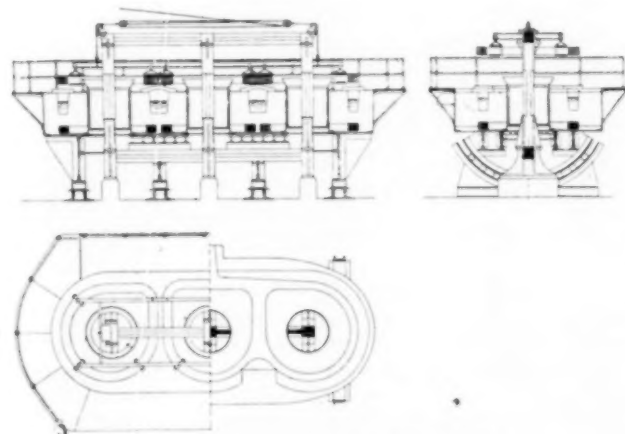


FIG. 2.—DESIGN OF 30-TON FURNACE

pay, seeing that the cost of the generator will thus be further increased, even if the output is reduced on account of the power-factor being improved.

It remains to figure out the necessary energy required for the furnace, the dimensions of the iron-core and the number of primary turns, together with the sectional area of these.

As the furnace will be used with a liquid charge, and no melting-down will take place, the amount of energy required is only the amount necessary to keep the bath at the required temperature. Apart from the small losses in the primary coil and the iron core, all the energy is converted into heat in the charge itself, and, as soon as a constant temperature is reached finally conducted to the surface of the furnace and given out from this by radiation, conduction and through the cooling arrangements. If water cooling is arranged in the same way and the same amount of water increased correspondingly, it may therefore be assumed that the loss of heat, and consequently the energy required to keep the charge hot, will be in proportion to the surface of the furnace.

According to the dimensions of the two furnaces the surface of the new furnace is between two and two and a half times as large as the old one. The amount of energy required in the old one to keep the charge in a molten condition during the night was approximately 180 kw., and during the refining the energy varied between 250 kw. and 350 kw. The minimum

energy required for the new furnace should consequently be $2 \times 180 = 360$, and the maximum, $2.5 \times 350 = 875$. He assumes 700 kw., but at the same time the generator should be constructed in such a way that, by merely altering the voltage it can give anything between 400 and 1,000 kw. With 700 kw the current per ring becomes approximately 46,000 amperes, and the current density in the bath will be about 0.34 ampere per sq. mm.

The further figures are then easily calculated.

The table below shows the principal data for the old furnace and the new one:

	Old Furnace.	New Furnace
Total capacity.....	5 tons	30 tons
Capacity per leg.....	2.5 tons	10 tons
Diameter of bath.....	2.1 m.	3 m.
Width of bath.....	20 cm.	30 cm.
Depth of bath.....	27 cm.	45 cm.
Total surface of masonry.....	55 sq. m.	110 sq. m.
Total length of platform.....	8½ m.	13 m.
Total width of platform.....	5¼ m.	6.5 m.
Sectional area of core per leg....	1800 sq. cm.	1200 sq. cm.
Weight of core.....	15 tons	23 tons
Number of primary turns per leg.....	15	13
Sectional area of primary windings	1000 sq. mm.	4000 sq. mm.
Weight of copper per leg.....	0.875 tons	ca. 4.5 tons
Total weight of copper.....	1.75 tons	ca. 13.5 tons
Energy used.....	250 kw.	700 kw.
Copper losses.....	12.3 kw.	40 kw.
Iron losses.....	18 kw.	18 kw.
Power-factor	0.65	0.50
Voltage	250	230
Periodicity (½ alternations per second) .	12½	8
Amperes per phase.....	1400	3540
Kind of current.....	1 phase	3 phase.

The number of primary turns of the new furnace is given above as 13, but in order to be able to run the furnace at full charge, even if the inductance and resistance should prove to be somewhat different from the values given, two turns should be added per leg, making the number of turns in the upper coils 8. The last three turns in each of these coils should be provided with terminals, enabling one to use at will 12, 13, 14 or 15 turns per leg.

The generator should, according to the foregoing, be designed for a normal output of 700 kw., 8 periods (corresponding to 16 alterations) per second, 230 volts, 3,540 amperes, and a power-factor of 0.5. At the same time it must be able to give 400 kw. at 165 volts, and 1,000 kw. at 275 volts, with the same power-factor, and 250 and 125 kw., respectively, with a power-factor of 0.362. The maximum current will thus be 4,300 amperes.

The engine or turbine driving this generator should be for a maximum output of 1,550 effective h. p., and run at a practically constant speed at all loads between 600 h. p. and 1,550 h. p.

This plant should be quite sufficient to supply energy to the 30-ton furnace, even if the constants of this should vary considerably from the figures given above. In order to study this, he has figured out what would happen, and the maximum and minimum energy which could be supplied to the furnace, in case either the resistance or the inductance or both should vary 25 per cent from the assumed value, either up or down. The results are given in this paper in form of a table.

Fig. 1 gives the designs of the old 5-ton furnace, Fig. 2 the design of the new furnaces.

In a communicated discussion Mr. Unger, of the General Electric Company, emphasized that the power installation as proposed in Mr. Hiorth's paper would be very costly on account of the use of so low a frequency as 8. He thought it would be cheaper to combine a number of smaller units in one furnace. In such a way it would be possible to build up an induction furnace of any size for commercial frequencies.

Dr. Waterhouse asked whether there would be any advantage of such a 30-ton pure induction furnace over a Roehling-Rodenhauser furnace or over an arc furnace of the same size.

Mr. FitzGerald replied to this question that the Roehling-Rodenhauser furnace for such sizes would be found greatly superior to other forms of induction furnaces, but made no comparison with any arc furnace.

Mr. Carl Hering referred to some calculations which he had made concerning the possibility of reducing heat losses on the basis of Mr. Hiorth's figures. The final conclusions will be published in an article in this journal. This was the last paper of the morning session.

An enjoyable lunch was then served in one of the halls of the University, and shortly afterward the second session began, being held as in the morning in the lecture room of the Chemistry and Mining Building.

Mineral Resources of Ontario.

The first paper was presented by Mr. T. B. GINSON, Director of Bureau of Mines of Ontario, on the mineral resources of Ontario. With the aid of a map the author sketched the situation and geology of the different portions of Ontario and gave an account of the mineral resources which have been found or may be expected to be found in these different portions. In Eastern Ontario there are a great variety of ores, but northern Ontario is the real home of the metallic deposits.

Mr. Gibson discussed at some length the production of silver, nickel, and gold in Ontario. There have been two eras of silver production in Ontario; the present one began with the opening of the Cobalt mines in 1904.

The nickel industry depends on the nickel-copper ores in the Sudbury district.

The gold industry is at present widely scattered and while it is not yet very successful it may be expected to become of great importance in the future.

Mr. Gibson concluded by suggesting two problems of importance which might be solved by electrochemical means. The first has to do with corundum which is practically pure alumina and is used at present mainly as an abrasive. But it is a richer ore of aluminium than bauxite and a process of making aluminium from corundum would be very profitable.

The second problem is the possible recovery of potash. There are deposits of feldspar in Ontario containing 13 to 14 per cent of potash. If the potash could be gotten out of it we would be able to get independent of Germany which has at present the monopoly in potash salts.

In the discussion Dr. Whitney remarked that another problem which might be worked out profitably is a process for the complex cobalt silver ores.

Addition Agents in Electrolytic Copper Deposition from Solutions Containing Arsenic.

A paper by Dr. CHING YU WEN and Dr. EDWARD F. KERN gives an extensive account of an investigation carried out at the School of Mines of Columbia University on "the effect of organic and inorganic addition agents upon the electrodeposition of copper from electrolytes containing arsenic."

Since the object of this investigation and the results obtained were already discussed at some length in our June issue, page 318, it will suffice here to give the chief final conclusion. The combined "addition-agents" of gelatine and sodium chloride, and of glue and sodium chloride, when present in small amounts, are the most suitable and most satisfactory "addition-agents" for copper sulphate electrolytes, especially such as contain a high proportion of arsenic, as with these the deposited copper possesses the greatest ductility and the highest purity. But the combined "addition-agent" of gelatine and sodium chloride (say 0.01 to 0.02 per cent. gelatine and 0.01 per cent. Cl as NaCl) seems to be a little better than sodium chloride and glue, for it gave smoother deposits. In the case of glue and inorganic compounds, the use of cuprous chloride,

cupric chloride or hydrochloric acid is just as effective as sodium chloride; but for economic reasons sodium chloride is to be preferred.

In a communicated discussion by Mr. W. L. Spaulding, of the Buffalo Smelting Works, reference was made to experiments on the addition of NaCl to copper refining solutions. In one case the addition of the salt had been found to have deleterious effects, causing a slimy layer on the anode.

Conductivity of Mixtures of Copper Sulphate and Sulphuric Acid.

A paper by Prof. H. K. RICHARDSON and Mr. F. D. TAYLOR of the Pennsylvania State College gave an account of a series of measurements of the conductivity of mixtures of copper sulphate and sulphuric acid.

In the absence of the authors it was presented in abstract by Mr. Lidbury.

To obtain figures for calculating the voltage drop due to electrolytic resistance in a copper refining tank knowledge is necessary of the conductivity of sulphuric acid and copper sulphate mixture. This was measured by the authors in different proportions at two different temperatures, namely, 25 deg. C. and 45 deg. C.

CONDUCTIVITY IN RECIPROCAL OHMS PER CM.										
25° C.					45° C.					
Grams H ₂ SO ₄ per 100 c.c.	0	5	10	15	20	0	5	10	15	20
CuSO ₄ ·5H ₂ O Grams per 100 c.c.	0	0.208	0.416	0.565	0.683	0	0.246	0.492	0.683	0.839
5	0.953	0.204	0.388	0.531	0.646	0.0205	0.242	0.461	0.643	0.791
10	0.0221	0.195	0.350	0.500	0.600	0.0294	0.222	0.422	0.606	0.738
15	0.0343	0.189	0.338	0.458	0.558	0.0468	0.217	0.381	0.545	0.690
20	0.0423	0.182	0.319	0.433	0.533	0.0574	0.212	0.378	0.521	0.643

In practical work most all strengths of solution are given in terms of percentages. To convert the values in the above table to these units the following table may be used.

Percentage		Grams per 100 c.c.		Gram Equivalent per 1000 c.c.	
Copper	CuSO ₄	CuSO ₄ ·5H ₂ O	Copper	CuSO ₄	CuSO ₄ ·5H ₂ O
1.00	2.5	3.91	1.04	2.56	4.0
2.00	5.0	7.82	2.09	5.25	8.2
3.99	10.0	15.64	4.41	11.05	17.4
5.99	15.0	23.46	6.97	17.50	27.4
6.98	17.5	27.37	8.37	22.00	32.9
	% H ₂ SO ₄		Grams per 100 c.c.	Grams Equivalent per 1000 c.c.	
	5		5.15	1.05	
	10		10.7	2.18	
	15		16.5	3.38	
	20		22.8	4.66	

Measurements were also made of the temperature coefficient of the conductivity.

Of the general conclusions of the authors these are the most important:

Addition of copper sulphate to solutions of sulphuric acid increases the conductivity of the mixture if the sulphuric acid is less than 3 grams per 100 cc. On the other hand, it decreases the conductivity if the sulphuric acid is over 3 grams per 100 cc. If the sulphuric acid is just 3 grams per 100 cc. the addition of a little copper sulphate to the solution of sulphuric acid has no effect on the conductivity of the mixture.

The temperature coefficient of mixtures of equal parts of copper and sulphuric acid appears to be that of the sulphuric acid alone.

Transformation of Other Forms of Carbon into Graphite.

A paper by Mr. W. C. ARSEM, of the Research Laboratory of the General Electric Company, Schenectady, N. Y., discusses the conditions under which "amorphous carbon" is transformed into graphite. These conditions have been the subject of much discussion, and many statements have found their way into literature which are not supported by experimental evidence.

The following theories are commonly held:

1. A high temperature alone will convert "amorphous carbon" into graphite (Moissan).
2. Pure carbon is not converted to graphite by heat alone (Berthelot).
3. Graphite is the result of intermediate formation and decomposition of carbides, due to the presence of mineral matter as an original constituent, or purposely added (Acheson). For instance, in one of Dr. Acheson's patents (617,979, Jan. 17, 1899)

it is mentioned that a mixture of 97 parts coke or charcoal and 3 per cent iron oxide can be changed to a greater or less extent into graphite by varying the time and temperature of heating. As the iron is insufficient to convert all the carbon to carbide, it is assumed to have a catalytic effect, first forming a carbide which decomposes, yielding graphite and setting free iron, which again forms carbide, and the cycle is repeated.

The present author has collected all available references relating to the problem. These are summed up in the paper.

He also emphasizes the ambiguity in the use of the term "graphite." The Brodie test is thought not to be decisive. The most characteristic property seems to be specific gravity. Mr. Arsem therefore proposes the following definition:

Graphite is that allotropic form of carbon having a specific gravity of 2.25 to 2.26.

Those varieties of carbon which have some of the physical properties of graphite, such as color, softness and streak, but a lower specific gravity, may perhaps be regarded as impure graphites; that is to say, mixtures of graphite with other forms of carbon.

Mr. Arsem then gives an account of an experimental investigation the object of which was to clear up these two points at issue:

1. Can a pure form of carbon be transformed into graphite by simply heating it to a high temperature?
2. If this is not the case, is it possible to cause this transformation to occur by heating the carbon, well mixed with a quantity of mineral matter, insufficient to form carbides with all the carbon present?

There are two ways of attacking the problem.

(a) To determine the effect of heating various forms of pure carbon alone and with small amounts of added mineral matter.

(b) To remove the mineral matter from certain varieties of impure carbon which ordinarily change to graphite when heated and see if the change will still occur.

Both methods were tried and the following results were obtained:

All the pure forms of carbon which have been tested, when fired above 3000°, reach a limiting density which is not appreciably raised by the addition of small amounts of mineral matter. The end product is graphite in some cases and not in others. Pure petroleum coke, heated without addition of mineral matter, is converted into graphite of excellent quality, while lampblack, although it increases in density, does not reach the value corresponding to graphite, nor acquire any of its other physical properties, even when heated with various oxides.

The impure carbons show a similar behavior. The properties of these carbons after firing are characteristic for each variety of carbon and independent of the amount of ash present.

Anthracite coal is only imperfectly graphitized by heating. The specific gravity of the fired material was approximately the same for three samples having a range of ash content. Moreover, coal from which most of the ash has been previously removed graphitizes better than the crude material.

Bituminous coal coke, which graphitizes fairly well, yields an even better product if a part of its ash has been removed before firing.

It must, therefore, be concluded that a small amount of mineral matter exercises no beneficial effect in the manufacture of graphite by the heating of carbon, and that the quality of the product cannot be improved in this way. As to the effect of mineral matter on the rate of conversion, no data are yet available, although some experimental work along this line has been started. At 3000° the maximum density is reached in less than 15 minutes for any variety of carbon.

Some theory of the nature of graphite and of amorphous carbon is needed which will permit a rational explanation of the changes which occur on heating. Mr. Arsem offers the following discussion as a tentative step in this direction:

Graphite, in the most restricted sense of the term, is an

allotropic form of carbon having a definite and not very complex molecular configuration. The molecule might, for example, be regarded as two benzene rings side by side and joined at all six angles, the extra bonds being satisfied as in the usual centric formula. Such a formula might be used to explain the formation of mellitic acid and graphitic acid by oxidation of graphite.

Amorphous Carbon. When an organic compound is decomposed, there results a mixture of substances constantly increasing in complexity until finally carbon is obtained. This carbon need not be regarded as a simple substance, but may be considered to be a mixture of many varieties of carbon, each with a different number and arrangement of atoms in the molecule. According to this view, it seems better not to regard "amorphous carbon" as the name of a distinct allotropic form, but rather as a general term covering all varieties of carbon except graphite and diamond.

In speaking of amorphous carbon, therefore, the source of each sample should be stated.

The Mode of Transformation. In a given sample of amorphous carbon, some of the molecules will be capable of easily undergoing rearrangement under the influence of heat to form graphite molecules, while others will not, and the proportion of molecules capable of such change will determine the character of the final product.

Petroleum coke would therefore consist almost wholly of graphitizable molecules, while anthracite coal contains a smaller proportion.

Mr. Arsem's paper which had been read in the absence of the author by Mr. Hansen was briefly discussed by Mr. Lidbury and Mr. Hansen.

Electric Laboratory Furnace with Resistor of Ductile Tungsten or Molybdenum.

A paper by Messrs R. WINNE and C. DANTSZEN of the Research Laboratory of the General Electric Company, of Schenectady, N. Y., describes two simple electric furnaces for laboratory work with resistors of ductile tungsten or molybdenum wire or ribbon. On account of their very high melting points and relatively low cost these metals are well adapted for that purpose.

The first type is a **crucible furnace** and consists of a helix of ductile tungsten or molybdenum, supported by an alundum tube and protected from oxidation by a hydrogen atmosphere.

Fig. 1 shows the furnace in vertical section. The heating wire, of ductile tungsten or molybdenum, is wound on *G*, an alundum cylinder which is molded plain inside and with a helical groove on its outer surface. *A* is a Battersea crucible; *B* is an inverted Battersea of which the bottom has been cut off; *C* is the crucible bottom, used as a cover. The bottom of *A*, as well as the space between *A* and *B*, is filled with powdered silica. Hydrogen is constantly supplied to the furnace through the pipe *D*, and burns as it escapes from the top of *B*. This gives a protecting atmosphere for both the tungsten or molybdenum winding and the object to be heated. Current is supplied to the winding through two large copper connectors *E*.

An object to be merely heated, as for annealing, may be inserted directly into the alundum cylinder. Material to be melted is placed in a small crucible, preferably made of pure alumina, and this is let down into the alundum cylinder.

In the apparatus shown in Fig. 1, the Battersea crucibles *A* and *B* are sizes *O* and *J* respectively. The alundum body *G* (made by the Norton Company, of Worcester, Mass.) consists of two cylinders placed end to end. Each is 7.6 cm. high and 5.1 cm. inside diameter, while the distance from one convolution of the helix to the next is .95 cm. The winding consists of 260 cm. of square molybdenum wire, 1.27 mm. on a side. This wire is first wound on a mandrel 3.8 cm. in diameter, and wire and mandrel are then heated up to about 800 deg. C. Upon then releasing the coil, it expands to the diameter of the alundum body, on which it can then be screwed. The copper connectors

E are 0.8 cm. in diameter, and the ends of the coil are simply clamped in with set-screws. The copper connectors and the hydrogen inlet tube are held in place in the crucible wall by a mixture of powdered silica and water glass, which at the same time prevents loss of hydrogen.

This furnace can be safely run up to 1700 deg. C. At this temperature it calls for 25 volts and 45 amp.

Pure iron can be melted and ferro alloys and other alloys can be produced in this furnace.

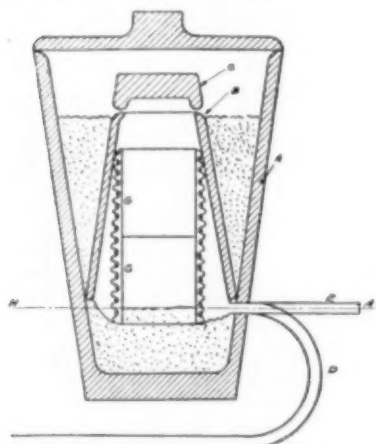


FIG. 1.—CRUCIBLE FURNACE WITH TUNGSTEN WIRE RESISTOR.

The second type is a **tube furnace** and consists of a porcelain or alundum tube, wound with tungsten or molybdenum foil. Around the tube is a tight metal casing filled with powdered silica, and, to prevent oxidation of the winding, a hydrogen atmosphere is maintained in the casing. Fig. 2 shows the furnace in vertical length section. *A* is an alundum tube, 2.3 cm. inside diameter and 46 cm. long. The casing *D* is made of thin sheet iron and is oxy-acetylene welded. The winding *B* is a molybdenum ribbon, 0.184 mm. thick, 2.54 mm. wide and 445 cm. long. *I, I* are heavy copper leads, fastened to the ends of the coil by twisting the latter about them. *C* is the powdered silica packing, *F, F* is some asbestos wool, used to prevent the escape of hydrogen at the ends of the casing.

H and *G* are the hydrogen inlet and outlet tubes respectively, *E* is a rubber stopper.

With this furnace a temperature of 1600 deg. C is readily attained. At this temperature it calls for 80 volts and 14.3 amp. The two furnaces described are adapted to heating the charge in an atmosphere of hydrogen. In case it were desirable to heat

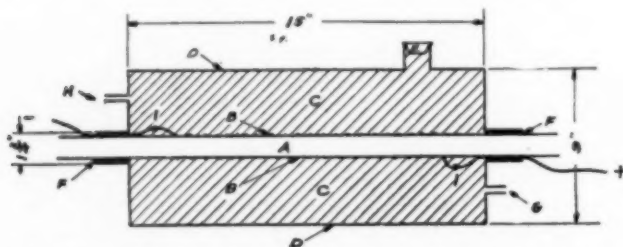


FIG. 2.—TUBE FURNACE WITH MOLYBDENUM RIBBON RESISTOR.

in an oxidizing atmosphere, it would be necessary to replace the very porous alundum by porcelain, as otherwise the furnace winding would become oxidized.

In the discussion of the paper of Messrs. Winne and Dantzian in which various speakers participated, Mr. Saunders suggested that the porosity of alundum tubes could be overcome and that their field of application could thereby be extended by providing a glazed inner surface. The melting point is high enough.

Electric Properties of Sodium, Potassium and Their Alloy

A paper by Dr. EDWIN F. NORTHROP, of Princeton University, gives new data on some electrical properties of sodium, potassium, and their alloy. The author intends to determine the electric resistivities and temperature coefficients of as many metals as possible at higher temperatures. To carry out

the investigation in a systematic manner, he will take up the elements in the order of Mendeleff's table. The investigation began with sodium and potassium, which fall in the first group. In view of their low melting points it seemed sufficient to limit the investigation to the temperature range between 0 deg. and 150 deg. C.

A glass tube, supplied with platinum potential and current terminals, was first filled with pure mercury, and the resistance of this, throughout the range of temperature employed, was determined between potential points. The same tube was then filled with the metal, in a molten state, which was to be measured. Its resistance, between the same potential points, was then measured at various temperatures as the metal cooled, from the highest to the lowest temperature. From the data so obtained the specific resistances and temperature coefficients were determined.

The methods of measurements are described in some detail.

For the resistance of the mercury in the interval from 20 to 110 deg. C the following relation giving the relation between resistance and temperature was found.

$$R_t = 0.02889 (1 + 0.0010036t)$$

where *t* is degrees C above zero.

Substance	Spec. Res. at 20° C. 10 ⁻⁹ ×	Spec. Res. at 100° C. 10 ⁻⁹ ×	Temp. Coef. Low Range	Temp. Coef. High Range	Density at 0°
Mercury.....	95,810	103,350	0.001003 (20° to 100°)	13.595
Sodium.....	4,875	9,705	0.0053+ (20° to 80°)	0.00494+ (100° to 140°)	0.978
Potassium.....	7,118	15,275	0.0058 (20° to 50°)	0.00507+ (66° to 120°)	0.87
Alloy, K+Na.....	33,800	37,795	0.00147 (46° to 125°)	0.92
Copper.....	1,721	2,266	0.00387+	8.89

Spec. res. of Na at 20° C. = 0.05088 × Spec. res. of Hg at 20° C.	
" " Na " 100° C. = 0.09390 × " " " 100° C.	
" " K " 20° C. = 0.07429 × " " " 20° C.	
" " K " 100° C. = 0.1478 × " " " 100° C.	
" " Na+K " 20° C. = 0.3528 × " " " 20° C.	
" " Na+K " 100° C. = 0.3657 × " " " 100° C.	

Some figures in this table are interesting. First the very high coefficient of potassium in the range 20 deg to 50 deg. of 0.0058 and the slightly smaller coefficient of sodium in the range 20 deg to 80 deg. of 0.0053. Potassium is thus probably unique in possessing the highest coefficient of any pure metal in this range that has been observed. It should be noted that the coefficient of the alloy of potassium and sodium, instead of being approximately the mean of the coefficients of sodium and potassium when liquid, is only between a third and a fourth of the mean value of the coefficients of its constituents in the same temperature range.

The mean specific resistance at 100 deg. of sodium and potassium is $12,490 \times 10^{-9}$, while the specific resistance of the alloy, in equal proportions by volume, is $37,795 \times 10^{-9}$, or slightly over three times as great as the mean specific resistance of its constituents. This is an interesting physical fact which may help to throw light upon the theory of conduction in metals.

In this connection we may note that alloying metals generally, if not always, increases their specific resistance and lowers their melting point.

The abrupt increase in resistance of the metals experimented with, expressed in percentage, is as follows:

Sodium, in passing from the solid state at 93.5 deg. to the liquid state at 96.3 deg., increased 44.8 per cent.

Potassium, in passing from the solid state at 60.4 deg. to the liquid state at 63.2 deg., increased 52.2 per cent.

The alloy of sodium and potassium, in passing from 2.8 deg., at which temperature it may probably be considered as completely solid, to 20 deg. C., at which temperature it may probably be considered as completely liquid, increased in resistance 48.5 per cent.

The ratio of the resistivity at 20 deg. C. of sodium to copper is 3.058, but the density of copper is almost exactly nine times that of sodium, which, in round numbers, makes sodium three times as good a conductor per unit of weight as copper. Hence it could compete with copper as a conveyor of electric power if it cost anything less than 3 times as much as copper per pound, provided, of course, current-carrying capacity were the only consideration. It has been suggested before now that sodium might be used in iron pipes in a practical way for carrying electric power. Dr. Northrup suggests that inasmuch as electrochemical processes often necessitate the carrying over short distances of enormous currents, the consideration of the proposal might be with advantage taken up afresh.

The paper was discussed by Messrs. Morrison, Whitney, Hering, and Lloyd. Reference was made to the considerable change in electric conductivity of steel at the melting point.

Electric Resistivity of Iron Alloys.

A paper by Prof. CHAS. F. BURGESS and Mr. JAMES ASTON, of the University of Wisconsin, discusses the electric resistivity of various iron alloys. In the design of electrical machinery, the electrical resistivity of the iron used for electromagnets is of an importance second only to the magnetic permeability, particularly for alternating-current working. A high resistance is desired, since it will reduce to a minimum the eddy currents set up by the reversals of the magnetic flux. It is

TABLE I.—HIGH ELECTRICAL RESISTANCE. RELATIVE ABOVE 5.

Composition	Microhms per cm.*	Relative Resistivity	Condition
10 Cr 5 Si	103.0	8.50	Forged
10 Cr 12 Ni	99.6	8.23	"
10 Cr 20 Ni 2 Si	93.0	7.68	"
10 Cr 5 W 2 Si	92.2	7.61	"
15 Cr 2 Si	92.6	7.55	"
7.23 Cr 9 W 3 Si	89.5	7.39	"
10 Cr 3 Si	87.5	7.23	"
10 Cr 2 Si 2 Ti	86.0	7.11	Quenched
13 Cr 1.8 Si	85.6	7.08	"
10.3 Cr 1 V 2 Si 0.3 C	84.9	7.01	"
10 Cr 2 W 2 Mo 2 Si 0.3 V	83.8	6.93	"
21.4 Cr 3 Mn 1 Si 1 V	82.0	6.77	"
28.42 Ni	82.0	6.77	"
35.09 Ni	81.1	6.70	"
20 Ni 10 Mo 1 C	79.6	6.57	Forged
10 Cr 10 Ni 2 Si	78.8	6.51	"
2 Si 10 V	78.9	6.51	"
5 Cr 6 Mo 10 Mn 10 Ni 1 C	78.0	6.45	Quenched
10 Cr 2 Si 0.3 V 0.6 C	77.4	6.40	"
6.2 Si	77.2	6.37	Forged
20 Ni 10 Mo	76.9	6.36	Quenched
17 Cr 14 Si	76.0	6.28	"
20 Cr 10 Ni 1 Si 0.3 V	76.0	6.28	Forged
10 Cr 2 Si	75.0	6.20	"
10 Cr 1 V	74.8	6.18	Quenched
10 Cr 10 Ni 1 Si	74.2	6.13	"
10 Cr 10 Ni 1 C	73.8	6.10	Forged
10 Cr 2 W 2 Si	72.8	6.02	Quenched
10 Ni 2 Si 1 C	70.3	5.81	Forged
2 Ni 12 Mn 6 Mo 0.8 Si	68.5	5.66	"
7 Cr 10 Ni	67.3	5.65	Quenched
10 Cr 8 Ni	68.4	5.65	Forged
20 Cr 3 Mn 1 Si	67.1	5.55	Quenched
10 Cr 10 Ni	66.9	5.44	Forged
26.40 Ni	65.5	5.41	Quenched
6 Cr 1.68 Si	64.4	5.32	"
25.20 Ni	64.2	5.30	Forged
25.20 Ni	63.2	5.22	Quenched
4.65 Si	62.2	5.13	"
10 Cr 2 W 2 Mo 0.6 C	61.6	5.09	Forged
8 Cr 1 Si	61.2	5.06	Quenched
10 Cr 10 W	60.7	5.02	Forged
10 Cr 2 W 0.3 V 0.6 C	60.4	5.00	Quenched

also of interest to know the conductivity of the various iron alloys, regardless of their magnetic properties, because of their possible utility in rheostats, resistance heating wires and the like.

In connection with the series of tests which the authors carried out on the various properties of electrolytic iron and its alloys, they determined the electrical resistance of the numerous bars used in the magnetic tests. They started with electrolytic iron as base material (the resistivity of which is taken as unity) and made up the various alloys by addition of the other elements.

It appears as a result of these tests that all three of the

elements of arsenic, silicon, and tin, which are beneficial to the magnetic properties of iron, have the desirable accompaniment of high electrical resistivity.

Among the materials adapted for high resistances, various combinations of iron with nickel, chromium, and silicon are most suitable.

The change of resistance due to heat treatment in general follows the changes of physical hardness due to the treatment. Annealing usually results in a decrease of resistance. Quenching is uncertain in its effect; usually there is an increase of resistance with physical hardening of the bar.

There are grouped in Table I, in the order of their relative resistances, all those iron alloys tested which showed a relative value as compared with the electrolytic iron of *five or upwards*. (The electric resistivity of pure electrolytic iron being taken as unity.) The highest resistance recorded is 8.50, reached by a bar with 10 per cent Cr, 5 per cent Si. Examination of the table shows 43 bars listed, all of which contain chromium, nickel or silicon, either alone or in various combinations.

The paper was briefly discussed by Messrs. Hansen, Lloyd, Northrup, and Watts. It was stated that the determinations of resistivities had been made at ordinary temperatures.

Before the conclusion of the Thursday afternoon session Mr. F. A. J. FitzGerald read the report of a special committee which had been appointed by the president to consider the possibility of setting a higher standard for the papers published in the Transactions of the Society. The report had been very carefully prepared and contained a series of valuable suggestions which had been approved by the meeting of the Board of Directors the night before. A committee was appointed to carry the suggestions of the report into effect.

Mr. Edward R. Taylor read a set of recommendations on principles of water conservation which he suggested to present at the coming National Conservation Congress in behalf of the Society. These recommendations were approved and endorsed.

Electrochemical and Electrometallurgical Developments in Canada.

The first paper of the Friday session was presented by Dr. S. DUSHMAN, of the University of Toronto.

He first gave a review of the electrochemical plants at present in operation throughout the Dominion of Canada. Their number is at present not very large.

In the extreme west in the Province of British Columbia, there is in Trail the electrolytic lead refining plant (using the Betts process) of the Consolidated Mining and Smelting Company of Canada.

The oldest electrochemical industry in Ontario is the manufacture of calcium carbide at the plant of the Willson Carbide Company at Merriton. Carbide is also made by the Ottawa Carbide Company on Victoria Island, in that section of Ottawa known as the Chaudière.

In the old Heroult electric furnace plant at Sault Ste. Marie, in which Dr. Haanel made his well known tests for the Government on the reduction of iron ore in the electric furnace, and which was later taken over by the Lake Superior Power Company for the manufacture of nickel pig from pyrrhotite, the company is now making ferrosilicon for its own consumption.

Ferrosilicon is also made by the Electro Metals Company, at Welland.

On the Canadian side of Niagara Falls there are plants for making cyanamide, graphite, and carborundum. The American Cyanamide Company commenced operations in January, 1910, with a 10,000-ton plant and is at present producing to its full capacity. The whole output is shipped to the United States.

The Norton Company has a plant at Chippewa for the manufacture of crastolon—another trade name for carborundum or silicon carbide. The raw materials used are two different

grades of coke (a metallurgical coke with 92 per cent or more fixed carbon and 5 per cent ash, and a petroleum coke, containing about 91 per cent fixed carbon with less than 1 per cent ash), a very pure silica sand, and sawdust. Two grades of product are manufactured, green and steel gray. At present 2000 hp are employed. The whole output of the plant, which is 5 tons per day, is shipped to the company's main works at Worcester, Mass., where it is used for the manufacture of different abrasive articles.

The International Acheson Graphite Company has also established a plant near Niagara Falls, Canada. Its output in 1909 was 513,436 pounds of artificial graphite.

No electrolytic alkali plant is at present in operation in Canada. The old Rhodian mercury-cell plant at Sault Ste. Marie was discontinued years ago and has not been opened again.

At the present time the Canadian Salt Company, of Windsor, Ont., is erecting a plant at Sandwich for the manufacture of caustic soda electrolytically, the Gibbs cell being used. It is expected that this plant will begin operations very shortly.

There is an electrolytic gold and silver parting plant at the Royal Mint in Ottawa. The gold bullion (after being assayed) is melted with silver in such proportion as to form an alloy containing 40 per cent gold, 56 per cent silver, and 4 per cent base metals. The alloy is cast into anode plates and used in a dilute solution of silver nitrate and a little nitric acid, and the silver deposited out. When nearly all the silver is removed a spongy mass of gold of the original shape of the anode plate is left. This is washed and re-melted and treated as anodes by the Wohlwill gold refining process.

Most of the electrochemical industries of Quebec are situated around Shawinigan Falls. The Northern Aluminum Company, a branch of the Aluminum Company of America, produces aluminium by the Hall process. The total capacity of the works is 25 tons of aluminium per day, about 500 men being employed when the plant is running at full capacity. The output in 1909 was over 3100 tons.

Carbide is made by the Shawinigan Carbide Company, by a continuous process.

The Electro Reduction Company, at Buckingham, manufactures phosphorus and ferro-phosphorus from apatite.

In the second part of his paper Dr. Dushman gave an account of the possibilities of future electrochemical and electrometallurgical developments in Canada, discussing the available raw materials, water powers, and transportation facilities.

As to mineral resources the greater portion of Canada is as yet unprospected, about one-third lies within the prospected area and two-thirds without, and the mineral resources have been developed (leaving out the Yukon) practically only in the territory lying fairly near the southern boundary. Even the portions of the country represented as being within the prospected territory must not be considered more than partially explored for minerals, as new discoveries are continuously being made within the supposed prospected areas.

At the present time the mining industry of Canada is second only to agriculture. In 1886 the total mineral production was slightly over \$10,000,000; in 1909 it had grown to \$92,000,000 and in 1910 to over \$105,000,000. In 1909 the total mineral production of Canada had a value of \$105,000,000. Of this \$49,000,000 was the value of the metallic products including \$17,000,000 for silver, \$11,000,000 for nickel, \$10,000,000 for gold and \$7,000,000 for copper. Among the non-metallic products coal leads with \$30,000,000.

The gold resources of Canada have only been skimmed so far. About 90 per cent of the silver produced in Canada in 1910 was derived from the rich silver deposits of Cobalt.

The ores of the Cobalt district are extremely complex, and the problem of treating these ores economically is still awaiting a solution. Under present conditions one of the pressing needs for the further promoting of mining and metallurgical

activity in Northern Ontario is the devising of suitable methods for the treatment of the Cobalt silver ores. Only about 30 per cent of the ore shipped from Cobalt in 1909 was treated in metallurgical works in Canada, most of this being high-grade silver ores, and the rest was shipped to a number of large companies operating in the United States. With respect to the content of nickel, cobalt and arsenic in the ores, the mining companies are paid only for a small portion of the cobalt content, and nothing for the nickel and arsenic, in fact, in certain cases the latter two are penalized.

Another interesting problem in connection with the cobalt silver ores, is what to do with the cobalt.

In the solution of these problems there is undoubtedly a field for the electrochemist.

With respect to iron much attention has been paid in Canada to the possibility of using electric furnace processes since Canada has immense iron ore deposits and water, powers. What has held back this development in the past is the very large capital investment which would be necessary.

Electric furnace experiments have also been made on a larger scale with respect to zinc in Canada, but have not led to commercial operation.

As to water powers in Canada, the total amount of power already developed and in process of development in British Columbia is 75,000 hp. The numerous powers off the coast would aggregate another 50,000 hp.

In Ontario the sources of water power are almost innumerable.

The Ottawa River alone would be capable of supplying over 300,000 hp, while the numerous falls on the Gatineau can supply over 200,000 hp.

In the province of Quebec the numerous tributaries of the St. Lawrence present remarkable power possibilities.

In New Brunswick only the St. John river has been exploited.

The Grand Forks Power Company is building a plant at that place to develop 80,000 hp.

With respect to transportation facilities, Canada in 1867, had 2278 miles of railroad, while it has at present close to 25,000 miles. The Canadian Pacific Railway was only completed in 1886, while there are now two other transcontinental railways in Canada.

The author concluded with the remark that in the great evolution which Canada is to undergo during this century the electrochemical engineer will be called upon to solve some of the most important problems.

Metallic Cerium.

A long paper by DR. ALCAN HIRSCH on "The Preparation and Properties of Metallic Cerium" gives an account of a very extended and elaborate investigation which required over three years' work at the University of Wisconsin.

The investigation was undertaken with a two-fold purpose. First, starting from as pure cerium salts as it was possible to obtain, it was desired to prepare the metal in quantities sufficiently large that its physical and chemical properties could be studied and a large number of alloys could be made. Second, by preparing metal from the unpurified rare-earth residues from monazite sand, obtained as a by-product in the incandescent gas mantle industry, it was hoped that alloys of commercial value could be obtained. In other words, whereas the application of these residues as oxalates or oxides is very limited to-day, their value might possibly be increased materially if a useful metallic or alloyed product be prepared from them.

A historical review is given of former work done on metallic cerium and then the methods developed for the preparation of the starting material are discussed in some detail. The general scheme followed was to use the mixed rare-earth oxides for experimenting on the exact conditions required for the preparation of the desired salt, and after the proper con-

ditions of the process had been determined, to prepare the pure cerium salt. Oxalates of the rare-earths, a by-product of the commercial extraction of thorium and cerium from monazite sand, were employed for these experiments. The oxides prepared by calcining the oxalates at a temperature of 750-800 deg. C. in a gas muffle furnace, analyzed 97.8 per cent. rare-earth oxides, of which approximately 49 per cent. was cerium dioxide.

The first salts required were the **anhydrous chlorides** (composition RCl_3). There are several hydrated chlorides of cerium known. The chief difficulty is in their dehydration since cerium chloride is similar to magnesium chloride in that when its solution is evaporated to dryness and calcined, HCl is evolved with partial decomposition of the chloride to oxychloride and oxide.

The author discusses a good many methods for the dehydration of chlorides of cerium. One method of obviating the difficulty would be to use a non-aqueous solvent. The carbide RC_2 may be prepared by heating the mixed oxides with powdered graphite in an arc furnace. Alcoholic HCl, made by passing dry HCl gas into cooled absolute alcohol, acts readily on the carbides. But the solution of the chlorides prepared by this method always contains more or less graphite, and is very difficult to filter because of its viscosity. The method is in general unsuitable for large-scale operation.

Resort was, therefore, again made to dehydration methods. It was found that by treating the mixed oxides with concentrated HCl, evaporating to dryness, adding more HCl and repeating the process a number of times, a product was obtained corresponding to one of the lower hydrates of the chlorides. But the difficulty is to get rid of the last molecule of water. In this process certain chlorides of the metalloids, especially those of phosphorus and sulphur, can be used as catalysers in the dehydration of hydrated chlorides. Experiments in this direction are described and finally a method of dehydration which may be carried out by HCl gas alone (without phosphorus and sulphur chlorides) was evolved which was used in the research. This is as follows.

The double cerium-ammonium nitrate from the fractionating vats, where it has been purified and separated from the other rare-earths by fractional crystallization, was used. The absorption spectrum of a concentrated solution of the nitrates showed only the faintest traces of the neodymium and praseodymium bands. The contents of a large bottle containing five kilograms of the double nitrates was divided into four portions and placed in fourteen-inch (35 cm.) porcelain evaporating dishes. The dishes were filled with concentrated HCl and placed on steam-heated sand-baths. Each dish was suitably protected from drippings from the hood by wooden covers raised a few inches above the tops of the dishes and supported on silica bricks.

Fresh acid was added daily as required and the evaporation continued for about three weeks, at the end of which time the nitrates had been completely converted into chlorides and all the ammonium salts had been expelled. The white solid was broken into lumps and heated on an asbestos-covered hot plate to convert it to the monohydrated chloride.

The final dehydration was carried out as follows (see Fig. 1). About 400-500 grams were placed in a ten-inch porcelain evaporating dish. A short-stem funnel about nine inches in diameter was inverted and placed over the salt. A layer of asbestos fiber was packed tightly around the outside edge of the funnel and plaster of paris set over this. A glass tube passing through the stem of the funnel and reaching to just above the layer of chloride conducted in the dry HCl gas, which was passed at a rapid rate over the heated chloride. The salt was stirred around from time to time by means of a long, thick glass rod. The heating was done gradually, and the progress of the dehydration could be watched through the glass funnel and its completion checked nicely.

The anhydrous chloride was placed, while still warm, in dry

glass vessels, and the corks paraffined in to exclude moisture from the extremely hygroscopic product.

The chloride prepared by this method melts to a clear, limpid liquid, and is suitable for electrolysis.

It was observed that the anhydrous rare-earth chlorides may be partially converted into oxides or oxychlorides if they were carelessly heated in the presence of air. The best method of melting the anhydrous chlorides without decomposition was found to be by heating a small portion in a covered crucible until a clear melt was obtained, and then introducing successive small portions of the chlorides to the melted portion. In this manner circulation of air is avoided, as the crucible is kept covered. A good way is to first melt sodium or potassium chlorides, or a mixture of these, and then to add small amounts of the rare-earth chlorides, keeping the crucible covered as much as possible. A small amount of oxide or oxychloride will "gum up" the entire melt, and for electrolysis this condition must be carefully guarded against.

Another salt required in large amounts was the **anhydrous mixed fluorides**, RF_3 . Lead lined apparatus are needed in the preparation.

After discussing several methods and their preparation the following method is given as the most satisfactory: About a



FIG. 1—APPARATUS FOR FINAL DEHYDRATION.

much of liquid syphoned off as was possible, and washed twice with hot water and several times with 95 per cent. alcohol. Only a few minutes' time was allowed after each washing for the precipitate to settle.

The fluorides were then placed in a lead-lined cage-centrifuge, and most of the remaining alcohol removed in this manner. Absolute alcohol was added and the solution evaporated to dryness without decanting. The absolute alcohol dehydrates the centrifuged fluorides very well. They were then dried at 100°, and the temperature raised to 200° at the end. About six kilograms of the fluorides were prepared in this manner.

Preparation of Metallic Cerium.—Two methods were tried; first thermal reductions and second electrolysis.

Thermal Reductions.—Magnesium and calcium shavings and aluminum powder were used as reducing agents on the mixed oxides. These experiments were conducted in electrically-baked magnesia linings placed in graphite containers. Carbon and silicon were also used to reduce the oxides, but external heat was applied to give the high temperature required. No metal was formed in these last instances but the carbides and silicides respectively. The carbide RC_2 is pyrophoric, but the silicide RSi_2 is not. Reductions using magnesium and calcium in excess gave alloys, many of which were pyrophoric.

In the case of these thermal reductions, either an alloy, a compound or a lower oxide of the rare-earths was obtained. Owing to the high heat of formation of these oxides, the failure of aluminothermics in the production of metal was only to be expected. The author believes that a study of the

equilibrium conditions of these reactions would form an interesting and valuable research and might result in the discovery of lower rare-earth oxides.

Preparation of Cerium of Electrolysis.—Electrolysis of the anhydrous chlorides was first tried but was found to involve a great number of difficulties which are recorded in some detail.

Electrolysis of anhydrous fluorides was then taken up. The anhydrous mixed fluorides melt at a temperature of a little over 900 deg. C. and dissolve the mixed oxides with avidity. The electrical conductivity of the molten fluorides is low, but increases rapidly with addition of oxides. The solution containing about 20 per cent. of oxides conducts very well.

Several runs of electrolysis are described. The principal difficulties encountered in the electrolysis of the oxides dissolved in the fused fluorides were:

1. Oxides must be added to the fluoride to make a sufficiently conductive solution, and the melting-point of the bath is very high.
2. At the high temperature of the bath, rare-earth metal and graphite combine to form carbide.

One of the principal difficulties in all previous electrolyses was the formation of carbides, which made the bath viscous and unfit for electrolysis. It was thought that if a non-carbon cell was used, the formation of carbide would be less likely to occur, as the only carbon in the cell would be the anode. On account of their corrosive action and high melting point the fluorides could not well be used in other than carbon vessels. It was, therefore, decided to try the electrolysis in a wrought-iron vessel.

Several experiments are described which show (1) that rare-earth metal could be produced by electrolysis of the chlorides in iron vessels; (2) that the use of an insulated cathode of graphite gave a lower current efficiency and resulted in some of the metal being converted to carbide; (3) that it was necessary to regulate the temperature of the bath so as to have the electrolyte well fused and yet well below the alloying temperature of iron and rare-earth metal; and (4) that control of the electrolysis could be maintained by judicious addition of small amounts of potassium fluoride, sodium chloride, and barium chloride.

The experiments in which the anhydrous mixed chlorides were used, were so successful that the method of electrolysis was applied to pure anhydrous cerous chloride. An account is given of several runs made in this way. A difficulty encountered was the removal of the material and electrolyte at the end of the run. This difficulty was overcome by using a vessel of the following construction. (Fig. 2.)

It consisted of a 3-in. (8 cm.) iron pipe screwed tightly into a 5-in. (13 cm.) iron flange, to which a bottom plate was attached by means of four large bolts and nuts. The electrolyte was cerous chloride containing a small percentage of sodium chloride, to which very small amounts of potassium fluoride and potassium fluoride-barium chloride mixture were added when required (about every half hour or so). The function of these additional salts has already been explained in detail.

The voltage was kept at its proper value, which should not be greater than 15 volts. The average current used was 200 amperes, for three hours. The electrolysis required attention about every twenty minutes; that is, the anode needed slight adjusting or the temperature had to be corrected. An ingot weighing 380 grams was obtained. The number of ampere hours was 600, and the ampere efficiency 36.5 per cent.

The electrolyte remained liquid during the entire course of the electrolysis. About 1½ kilograms of cerous chloride was used. The cell held perfectly, and at the end of the run the electrolyte and metal were easily removed. This type of electrolytic vessel is highly recommended for similar work. Care should be taken that the flange and bottom plate are well faced and that the lap is wide enough (at least 1½ in. (4 cm.)).

Two methods of analysis of cerium are then described, one being gravimetric, the other volumetric.

The chief impurities of cerium (amounting in the aggregate to about two per cent.) obtained by the method previously described consisted of iron, cerium oxide and cerium carbide, of which over one per cent. was iron.

The best method of **purification of cerium** consists in preparing cerium amalgams. The amalgams are prepared by boiling

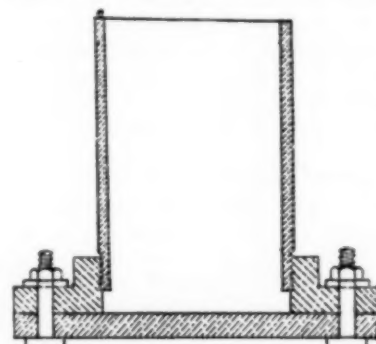


FIG. 2.—ELECTROLYTIC CELL.

ing mercury with cerium in a long iron pipe arranged with a condensing tube at the top. Solid amalgams are easily obtained by this method. The iron and impurities float to the top and may be skimmed off. The amalgams carefully prepared give only a very slight test for iron. The cerium and mercury may be separated by distilling the latter,

but this must be done in a high vacuum to prevent oxidation of the cerium. A high temperature is required to drive off all the mercury. The amalgam should be placed in a magnesia vessel, and this in turn heated in an evacuated quartz vessel. The high temperature required would cause the collapse of an evacuated glass vessel.

Physical Properties of Metallic Cerium.—In the determinations of the physical properties, metal of approximately 98 per cent. of cerium was used, unless otherwise stated.

Atomic weight 140.25.

Specific gravity 6.92 at 25 deg. C.

Hardness. For the determination of hardness, the scleroscope was used. The hardness of cerium varies, depending upon whether the surface of the metal is rolled, freshly cut, or old. The average value of the hardness for rolled surface was 25.9. The average for freshly cut surface was 9.5. The reading 100 is the hardness of the standard steel test plate.

Tenacity. Cerium is malleable and highly ductile. A strip of cerium was rolled to a thin sheet of a thickness of 0.015 mm. Cerium metal can easily be cut with a knife or scissors, and can be machined fairly well, although there is some tendency for the metal to buckle, as does lead.

The specific electric resistance of a sample containing 93.6 per cent. cerium and 4.5 iron was 71.6 microhms per cm. cube.

Magnetic properties. Pure cerium is weakly but definitely paramagnetic.

Melting point 635 deg. C.

The ultimate strength was determined in an Olsen testing machine. The average diameter of the test bars was 0.212 in. (0.55 cm.) There was no elongation with constant load at 350 pounds (159 kg.). The test bar broke with a snap (like cast iron) at 495 pounds (225 kg.). The ultimate strength is equal to 12,900 pounds per square in. (9 kg. per sq. mm.).

The specific heat of cerium (20 to 100° C.) was found to be 0.0524. The specific heating of copper is 0.0925.

The heat conductivity seems to be fairly high and, if so, cerium is probably one of the exceptional metals which has a low electrical conductivity and high thermal conductivity.

The latent heat of fusion seems to be fairly high.

Heat of oxidation. The average value of the heat of oxidation was 1,740 calories per gram, or 60,900 calories per gram equivalent ($\text{CeO}_2 = 243,600$).

Single potential. As the single potential of cerium could not be measured successfully, it was calculated and the single potential of cerium in normal solution of its chloride measured against the normal calomel electrode was thus found to be -3.16 volts.

Thermo-electromotive force. With a hot junction at 100°C and a cold junction at 30°deg. , the thermo-emf was 0.000250 volt. With the hot junction at 200.5°deg. and the cold junction at 30°deg. C. the thermo-emf was 0.000502 volt. The current flowed at the hot junction from copper to cerium.

Optical properties of magnesium cerium alloys were also determined.

Chemical properties of metallic cerium.—Cerium is very slightly attacked by cold water, but in boiling water a slow evolution of hydrogen gas occurs and the metal is tarnished black. At room temperature the following solvents have no action on cerium: Ethyl alcohol, amyl alcohol, chloroform, carbon tetrachloride, concentrated sulphuric acid, concentrated ammonium hydroxide, concentrated sodium hydroxide. Ethyl ether has a very slight action on the metal, as have also 3 per cent and 30 per cent hydrogen peroxide at room temperature. The action of dilute sulphuric acid, concentrated and dilute hydrochloric acid, concentrated and dilute nitric acid, in the cold, is moderately vigorous. Ammonium chloride and potassium chloride, at room temperature, have moderate action on the metal. This may be explained by the fact that in water a small amount of cerium hydroxide forms which is soluble in potassium chloride and ammonium chloride, thus causing fresh surface of the metal to be exposed and resulting in moderate attack of the metal by the solvent.

At boiling temperature the following solvents did not attack the metal. Chloroform, carbon tetrachloride, concentrated sulphuric acid, concentrated ammonium hydroxide. The metal was only slightly attacked by the following solvents at boiling temperature: Ethyl alcohol, amyl alcohol, ethyl ether and concentrated sodium hydroxide. Dilute nitric acid, ammonium chloride, potassium chloride and 3 per cent. hydrogen peroxide at boiling temperatures gave moderate action. Dilute sulphuric acid, concentrated and dilute hydrochloric acid, concentrated nitric acid and 30 per cent. hydrogen peroxide gave vigorous evolution of gas at boiling temperature.

The action of various gases on cerium was studied. A small amount of cerium filings was placed in a porcelain boat in a hard glass tube. The air was displaced by the gas under test, and the tube heated gently by means of gas burners. The cerium filings heated in chlorine emitted a very bright light at a temperature of 210° to 215°C. The salt formed was tested and found to be anhydrous cerous chloride.

In bromine, cerium burned at a temperature of 215° to 220°C. with emission of less light than in the case of chlorine. The salt formed in solution in water, and was identified as the bromide.

In iodine vapors no light was emitted, although the tube was heated to a temperature of 300°C. On examination of the contents of the tube it was found that some iodide had formed.

Cerium burned with luminescence when heated in air to a temperature of 160°C. If a lump of cerium is sealed in a glass bottle and kept warm for some time, a black powder is seen to form upon the surface of the cerium, and when the bottle is opened this powder ignites at room temperature. This is probably due to the formation of a highly pyrophoric suboxide of cerium.

When cerium filings are heated to 345° in hydrogen gas, the hydride forms without emission of light. The hydride has been described and studied by Muthmann. (*Liebig Ann.*, 325, 281.)

Cerium filings were heated in nitrogen to a temperature of 1000°C. No luminescence was observed, but some nitride was formed. This nitride is a black powder, and has been studied and described by Matignon. (*Comptes Rendus*, 131, (21), 837).

When the nitride is heated in potassium hydroxide solution, evolution of ammonia gas occurs. When exposed to air the nitride gradually changes over to a brown oxide.

Cerium filings were heated in the vapors of sulphur. When the black substance which is formed is treated with dilute sulphuric acid, sulphuretted hydrogen is evolved. Cerium sul-

phide has also been formerly studied by various investigators.

Cerium filings were heated to a temperature of 500°C. in an atmosphere of carbon monoxide. There was no visible reaction, but part of the filings appeared oxidized.

Alloys.—A number of different alloys of cerium were studied. In most cases the alloys were high in cerium, usually containing about 70 per cent. cerium. Alloys were made with silver, gold, platinum, copper, tin, antimony, arsenic, carbon, silicon, sulphur, selenium, tellurium, lead, calcium, sodium, aluminum, zinc, cadmium, chromium, manganese, iron, nickel, tungsten, mercury and magnesium.

Cerium and silicon form the silicide CeSi_2 . This compound may be formed by the reduction of the oxide of cerium. Large amounts of the mixed silicides have been prepared by the author. The following mixture is recommended for their preparation:

MnO_2	1,000	grams
Powdered graphite	200	"
Powdered silicon	450	"

The silicide of cerium is somewhat brittle, and may easily be pulverized to a fine powder. It is a splendid reducing agent. When the silicide is added to cerium so that the silicon content is about 15 per cent., a good pyrophoric alloy is obtained.

The iron-cerium alloys are very interesting, as they were the first pyrophoric alloys known, and were discovered by Dr. Auer von Welsbach. The alloys of about 70 per cent. cerium content are fairly hard and somewhat brittle. The microstructure of one of these alloys is shown in Fig. 11. It is seen that the structure is heterogeneous, probably consisting in part of a cerium-iron compound. These alloys have been the subject of some discussion in the literature,²⁰ as have the reasons for the pyrophoric properties.²¹ The writer believes that the question of the pyrophoricity depends upon the following factors: Cerium alone is soft and malleable, and when scratched with a file, small particles are not broken off. However, when the alloy is hard and brittle, small particles are easily detached from the mass, and the friction is sufficient to raise the temperature of these small particles to the incandescent point of cerium (160°). As metallic compounds (such as Cu_3Sn) are as a rule hard and brittle, the addition of such compounds to cerium usually assures a pyrophoric alloy. The alloy should be high in cerium, so that ignition occurs at a low temperature, and should contain excess of cerium above that required for the formation of a compound (such as CeSi_2) to act as a binder and so prevent the disintegration of the alloy, as in the case of the cerium-aluminum alloy containing about 60 per cent. cerium.

The whole series of magnesium-cerium alloys were prepared and investigated for their optional properties. The alloy containing about 83 per cent cerium is highly pyrophoric. Most of these alloys are brittle and can be easily pulverized. Excellent flashlight powders can be prepared from the ones of higher cerium content. These alloys also form splendid reducing agents, as the combination of cerium and magnesium is an endothermic reaction, and when the alloy is oxidized, more heat is emitted than from an equivalent mechanical mixture of the two constituents. The fact that the alloys of from 60 to 75 per cent. cerium content may be easily pulverized in a mortar to a fineness of 200 should render these alloys valuable for thermal reductions. A small amount of metallic vanadium was prepared by reducing V_2O_5 by this method. The alloys of from 75 to 85 per cent. cerium content can be pulverized, but they are so highly pyrophoric that it is difficult to prevent igniting them.

Dr. Hirsch had quite an interesting series of samples which were passed around. In the discussion of the paper he was highly complimented on his success by Dr. Whitney, Dr. Parsons, and Dr. Northrup.

Dr. Richards spoke briefly on the cause of the sparking of the alloys. The pyrophoric alloys are brittle. When scratched with a knife fine particles are given off which are

heated to incandescence, but with pure cerium which is soft this does not take place. Dr. Hirsch said that to get a pyrophoric alloy there must be enough cerium in it, generally above 70 per cent.

Mr. Hansen thought the scleroscope was a good instrument for testing the hardness of different steels among themselves, but not for comparing the hardness of different metals or alloys with each other.

He thought the scratch test would give better results for comparison.

Thermal Conduction and Convection in Gases at High Temperatures.

A paper by Dr. IRVING LANGMUIR of the Research Laboratory of the General Electric Company, deals with "thermal conduction and convection in gases at extremely high temperatures."

The author first describes a series of experiments on the formation of nitric oxide from air in contact with a glowing Nernst filament. Some theoretically curious results were obtained. They seem to suggest that some new and previously unsuspected phenomenon was taking place. A dissociation of the gas molecules into atoms or an absorption of energy by the emission of ions from the hot surface suggest themselves as possible causes, but this could only be decided by further experimental investigation.

However, the Nernst glower is not suitable for such experiments, because the range of temperature is limited. The author, therefore, continued his experiments with ductile tungsten wire in a hydrogen atmosphere. Some of the results obtained are as follows:

The energy required to heat the wire to a given temperature is very nearly independent of the size of the vessel containing the hydrogen.

The relation between energy consumption and temperature was determined, and the loss of energy was found to increase very rapidly with increasing temperature.

Up to about 1300 deg. K. the energy increases proportional to a power of the temperature between the first and the second, but above that it gradually increases more rapidly until at 3400 deg. K. it is actually increasing with more than the tenth power of the temperature. In a vacuum the energy loss is proportional to the 4.8 power of the temperature, and this exponent is constant over a very wide range. So, for a wire of about 0.045 mm. diameter, heated in hydrogen, the ratio of radiated to total energy reaches a maximum at about 2600 deg. K. At this temperature, however, only about 9 per cent of the total energy is lost by radiation, while 91 per cent is carried away by conduction through the hydrogen. At 1000 deg. K. the proportion of radiated energy is about 0.7 per cent, and at 3400 deg. it is 3.4 per cent of the total energy.

These results with hydrogen strongly confirm the conclusions drawn from the experiments with the Nernst glower. The most reasonable explanation of the very large heat conductivity of these gases at these extremely high temperatures seems to be that the molecules dissociate into atoms close to the filament, thereby absorbing a large amount of heat. The atoms then diffuse away from the wire, and there, because of the lower temperature, recombine and give up the heat that was absorbed from the wire.

Experiments are now in progress to determine the energy loss of a tungsten wire heated in monatomic gases. (From remarks made by Dr. Langmuir during the discussion it would appear that recent investigations of this kind by the author have made the correctness of the above explanation at least questionable, although the results are not yet conclusive.)

In connection with these experiments it should be noted that the candle-power measurements indicate that the melting point of tungsten is at least 3450 deg. K. Pirani had found 3570 deg. by a very unsatisfactory method of extrapolation, but the best value up to that time, and the one that even now seems to be

generally accepted, is that of von Wartenberg, who gave 3170 deg. K.

In the course of the discussion in which Dr. Northrup and Dr. Lloyd participated, Dr. Langmuir discussed the discrepancy of a tungsten filament from a thermodynamically black body and the advantages of determining the melting point of tungsten in hydrogen.

Measurement of Small Gas Pressures.

A paper by Dr. C. F. HALE, of the Research Laboratory of the General Electric Company, deals with the "measurement of very small gas pressures." The extension of studies of the phenomena in rarified gases to the region of very low pressures has made increasingly important the development of manometers whose indications of pressure are not dependent upon the use of mercury.

The manometer of Pirani (*Ber. der Deutsch. Phys. Ges.*, 1906, p. 686), depends for its ability to measure pressures upon the fact that at low pressures the heat conductivity of gases is a function of the pressure. It is only necessary, therefore, to observe the effects of varying heat dissipation, with pressure changes, upon some source of heat within the vessel undergoing evacuation.

Pirani has accomplished this by connecting to the system an ordinary tantalum lamp. The wire in this lamp was heated by a constant-voltage current. As the pressure in the system was decreased more and more, the heat lost by conduction through the gas was continually decreased; consequently the temperature of the wire mounted steadily, and, with the temperature, the resistance of the wire increased, thereby cutting down the current.

One had then only to calibrate ammeter readings against the indications of some form of manometer.

A better instrument was obtained when the tantalum wire was replaced by platinum, and the wire was clamped tightly to the anchor wires in order to keep constant the heat loss through the supports.

In simplicity both of construction and of operation, involving only current and resistance measurements within the range of instruments readily accessible, the Pirani manometer recommends itself at once.

However, in point of sensitiveness to pressure changes it is somewhat disappointing.

The present author has modified the Pirani manometer so as to increase its sensitiveness quite materially. The construction of the improved manometer is described in great detail and it is found to give reliable measurements of pressure down to 0.00001 mm. of mercury.

A method of calibrating the manometer is also described. Since the heat conductivity varies with the nature of the gas, it is necessary to calibrate the manometer with the same gas with which it is to be employed subsequently. Calibration curves are given for pure dry air and for hydrogen.

The Electrochemical Industries of Norway.

Prof. JOS. W. RICHARDS, of Lehigh University, who had just returned from abroad, gave a description of a number of electrochemical plants in Norway, which he had visited this summer. There are in the whole thirteen electrochemical plants in Norway and he visited nine of them.

First an aluminium plant at Kristiansand was noticed which operates with a capacity of 12,000 hp by the Heroult process producing 800 to 1000 tons of aluminium per year.

The next plant dealt with was a nickel refinery depending for its ore supply on a copper-nickel ore mine. The ore is smelted by semipyrritic smelting near the mine and the nickel-copper matte shipped to the refinery where the nickel and copper are electrolytically separated by the process of Hybinette (our vol. IV, pp. 33 and 34, 1906).

The next plant described was the experimental plant of the Badische Company for the manufacture of sodium nitrite, 200

hp being used for this purpose. The well known Schoenherr process for burning the atmospheric nitrogen by electric discharges is employed. But the company finds it more profitable to work up the nitrogen oxides into nitrite instead of nitrate. The nitrite is shipped to Germany. The production is 300 to 400 tons per year.

The next plant visited by Dr. Richards was the aluminium plant erected by the British Aluminium Company in 1906, using 2000 kw and producing about 600 tons a year.

Close by is a sodium plant which was started by Mr. Ashcroft. When Dr. Richards visited it it had been closed down, due to lack of water, but when the power supply does not fail, it is in operation. The Castner process for making metallic sodium is used with a few modifications.

At Odda there are two large and interesting plants, one being a carbide plant and the other a cyanamide plant.

The carbide plant contains 12 furnaces of 1500 hp each and one large one of 2000 hp. The factory works very regularly.

An interesting device for protecting the workmen against the heat from the furnaces is used there. It is a double screen of wire, two screens being 1 in. from each other placed before the furnace. The view is not shut off thereby, since one can look through the screen as through a mosquito screen, but it cuts off a great deal of heat.

Twenty thousand hp are utilized at these works, producing 30,000 tons of carbide per year. Of these 10,000 tons (largely fine stuff) are supplied to the cyanamide works. The carbide plant is to be doubled next year and the whole extra production is to be sent to the cyanamide plant.

This indicates that the cyanamide industry is in a good condition, but it is noteworthy that they prefer not to sell cyanamide itself as a fertilizer but to convert it into ammonium sulphate. The cyanamide is first converted into ammonia and then into ammonium sulphate. The sulphuric acid necessary for this process will be supplied from a near-by plant, to be mentioned below.

Four miles north of Odda there is the Tysse water power, where 125,000 hp will be available.

One of the interesting developments there is electric zinc smelting. Mixed Broken Hill concentrates will be used. They will be roasted and the sulphuric acid will be shipped to the cyanamide plant mentioned above. The zinc oxides will be reduced in electric furnaces. The furnace to be used is stated to be a modified De Laval furnace, but is probably a resistance furnace rather than an arc furnace.

Smelting of pig iron in the electric furnace has not yet been started in Norway, but will be started next year at Hardange. The design of the two furnaces, each of 4000 hp, will be based on the experience obtained at Trollhättan. Also, steel furnaces of the Ludvika type are installed there.

At Arendal there is another company which will produce pig iron in two 4000-hp electric furnaces. This is one of the undertakings of Mr. Eyde.

With respect to electric steel the Norwegians are looking now toward the use of steel scrap. At Stavange there is a company which breaks up old ships. The scrap is to be used by the Stavange Electric Steel Works. They estimate they will have a supply of 3000 to 6000 tons of scrap per year. The water-power plant is being built and the steel plant is to be completed by December.

The steel is melted down in a Siemens open-hearth furnace and roughly refined and is then still further refined in a Roehling-Rodenhauser electric furnace. There is also a rolling mill and a foundry.

At Mr. Hiorth's plant the induction furnace is used for melting down high-quality metal and making a high-quality steel, but he also uses an electrode furnace for working up the scrap.

Dr. Richards attended the jubilee of the University of Kristiania, and found that all Norwegians were extremely inter-

ested in the electrochemical industry, as they expected that electrochemistry would play an important part in the future development of Norway. Since strong restrictions against foreign capital have been put in force in Norway, Norwegians now are putting their own money into such developments. The King of Norway is much interested in the use of the electric furnace for iron and steel. Similar interest is taken in all these developments in Sweden.

Among the plants which were not visited by Dr. Richards is the large Nottoden plant for making nitrates from air by the Birkeland-Eyde process. It has been found that for large units the Birkeland-Eyde furnace, being compact, is more economical than the Schoenherr furnace, which must be built high up in order to increase its size.

As to the cost of power in Norway, Dr. Richards stated that power could be bought there from a power company for \$6.50 per hp-year. This is at a place right at deep water, so that shipping facilities are also provided. The cost of producing power was given as between \$4 and \$5 per hp-year.

Galvanizing Wire in Zinc Dust.

A paper by Mr. ALFRED SANG, of Paris, France, emphasizes that a perfectly clear conception of the forces at work is essential for successfully operating his process of galvanizing steel wire by heating it electrically in zinc oxide or zinc dust.* Once the nature of zinc dust was well understood and the action which different temperatures may have on it was clearly demonstrated, the application of the process was found to be, after all, not nearly as complicated as might have been feared, in fact it was less complicated than the other two processes in general use.

It may be recalled that zinc dust does not melt when heated to the temperature of fusion of zinc, but at higher temperatures reducing agents will "precipitate" the metal, as it were, and it can also be squeezed out under mechanical pressure. This property of resisting fusion is supposed and is probably due to the surface of each particle being formed of zinc oxide or zinc carbonate.

If zinc dust is heated evenly, the temperature is found to rise steadily until around 390° C., when the rise becomes much more rapid, attaining its maximum rate at about 420°, then falling gradually until in the neighborhood of 470° it again becomes normal. When the dust is cooled down an opposite effect is noted between 450° and 260°. If the operations of alternately heating and cooling the same dust are repeated, the exothermic and endothermic effects are manifested each time between the same temperatures, with their apices at about 420° on the rise and about 350° on the fall.

It is noteworthy that the exothermic effect on the rise occurs at the melting point of zinc metal, but when zinc is melted the effect is the very reverse, heat being absorbed at the moment of fusion to the extent of 28 calories per kilogram.

This represents, therefore, an anomalous critical point, and the author considers separately the operations of galvanizing in zinc dust below and above that critical point.

If a clean piece of steel wire is immersed in zinc dust which is heated to a temperature below 420° C., or at what is known as "sherardizing temperatures," a slow cementation takes place and zinc is absorbed from the very moment that the wire reaches the temperature of the dust; there is at the same time a slow sublimation of zinc at the surface increasing with time but sufficient for all practical purposes after a few hours of treatment; in the presence of reducing gases the time of treatment can be considerably lessened, but the penetration of zinc is, of course, less. This process is sherardizing, unsurpassed for many purposes, unsuitable for many others. There are two varieties of sherardized coatings; the gray, porous and brittle variety, mostly composed of zinc carbonate which darkens with age; the other variety has a high metallic luster

*This journal, Vol. VII., p. 532. Concerning sherardizing see this journal, Vol. V., p. 187; Vol. VI., p. 189, and Vol. VII., p. 485.

and is compact and of a finely crystalline structure. If this were the class of sherardizing produced by all existing plants, there would not be much hot-galvanized work produced outside of wire and sheets.

When we work with the dust at a temperature above the critical 420° , results are entirely different; galvanizing is instantaneous, taking place the very instant that the wire reaches the temperature of the dust. There is very little penetration of zinc in the iron; there may be vaporization and condensation of zinc as in sherardizing, although this is doubtful. A wire heated to 800° or 900° in a non-oxidizing flame and quickly thrust into cold zinc dust will condense around itself a considerable thickness of zinc. The best working temperature ranges between 500° C. and 600° C. The galvanizing is bright and more pleasing in appearance than that obtained by any other means.

The author does not go into details of the furnace used, which merely consists of a zinc-dust trough between two sand troughs provided with a set of cast-iron cover-plates over the zinc trough and power-driven rolls. As high a speed as 850 ft. per minute can be used for securing good merchant galvanizing. It is found preferable to run at 35 ft. only, running a large number of strands and attaching the coils end to end.

The process is economical as regards current consumption which outside of zinc and labor, with a very small amount of power for driving the rolls, represents the bulk of direct cost.

The dust very rapidly assumes a fairly constant temperature. When this is the case the consumption of electrical energy in watt-hours is $7/6$ times wst , where w the weight of wire treated (say 1000 kg), s the specific heat of steel $=0.18$ at 550° C. and t the temperature to be maintained, say 550° C. The calories needed are wst , hence by conversion into electrical units we get $7/6 wst$ watt-hours.

Figuring 75 per cent efficiency for the dynamo or transformer supplying the current, the above formula gives 154 kw-hours per long ton of wire of any diameter.

While the wire is softened by the process, a full annealing can only be secured by using a higher temperature, the consumption of electrical energy being then increased from 40 per cent to 50 per cent and the sand trough through which the wire passes after leaving the zinc dust must be appreciably lengthened. If a slight scale is to be removed, the high temperature must be used in connection with a reducing agent. Sand must be used with the zinc dust to prevent caking and cavitation in the path of the wire. Aluminium contact rolls and special mercury contacts are the only ones suitable for this kind of plant.

The wire can be brought out into the air immediately after coating, but it is preferable to let it cool by passing it through sand which brightens it at the same time.

The instantaneous galvanizing of steel in zinc dust above 420° is rapidly explained by completing the theory which the author developed as far back as 1907; it is now stated in its complete form as follows:

"Zinc dust is produced by the sudden cooling of the vapor of zinc and the transition from the gaseous to the solid state is so rapid that, for all practical purposes, the liquid state is skipped. The setting of the zinc, brought about by sudden chilling, prevents regular arrangement of the molecules along determined stereochemical lines; it forbids the formation of crystals for which time is required. The heat of solution of matter in an amorphous state is always higher than when in a crystalline state; the same is true of a strained condition which likewise presents an amorphous appearance. At the time of formation considerable heat is retained which would have been radiated by slow cooling. The condition is one of physical instability resulting in readier and, oftentimes, explosive decomposition.

"The physical instability of zinc dust may be explained thus: In the prolongs of zinc furnaces the uncondensed vapor is suddenly chilled and collects in minute drops which are instantly congealed at the surface and oxidized, forming a rigid

spherical skin or crust. Within this crust the liquid zinc cools and contracts more slowly but the rigidity of the crust does not permit the drawing in of the surface to form the facets of a crystal and compensate this internal contraction. Thus voids must exist within the crust and we may presume a structure which might be described as 'piped radially' or as negatively crystalline.

"Below the melting point of zinc this condition persists, but affinities and thermal inducements find a readier response from zinc dust than from any other form of zinc, both by reason of the enormous amount of free surface and because of this unstable condition.

"At about 415° , which is the melting point of zinc, the strain within the zinc dust particle is totally or almost totally relieved as indicated by the loss of heat, although some of the latent heat must be retained for fusion.

"At high temperatures zinc dust is pasty, because formed of innumerable little bags of melted zinc, but these will not coalesce on account of their small size and closely packed condition, unless considerable pressure is used. These minute bladders might be compared to soft-shell turtle's eggs; they form a plastic and cellular mass.

"If, however, a reducing agent, say a drop of coal tar, is introduced, reduction of the skin takes place and we get a small mass of more or less spongy metallic zinc. If a metal which is electronegative to zinc, such as iron, is put into contact with these bags of zinc, reduction and an exchange of metals take place, the bag is punctured at the point of contact and the zinc spreads itself on the iron.

"Hence it is that wire can be galvanized in this manner by means of a plant closely resembling a hot galvanizing plant, for the process is, in fact, one of hot galvanizing."

Electric Nickel Smelting.

A paper by Mr. WALTER LEONARD MORRISON, entitled "Electric Furnace Treatment of Nickel Ore and the Development of a Commercial Process" (being a thesis presented to the Case School of Applied Science), gave an account of the work done by the Consolidated Nickel Company, at Webster, N. C., about which an article was already published in this journal, Vol. VIII, p. 277 (May, 1910). This paper was distributed on the train on Saturday.

The nickel deposits are located about one-half mile from Webster, N. C. The analysis of ores from the same cross cut vary greatly in chemical composition and color. The elements which vary most are iron, magnesia, and alumina.

The reddish brown variety of ore carries the greatest amount of iron and alumina, while the green ores carry the greatest amount of magnesia.

A typical analysis of the hard green ore is 1.90 per cent nickel oxide, 34 silica, 48 magnesia, 3.4 alumina, 1 iron oxide, 1.7 chromic oxide, 10 water of crystallization.

A typical analysis of the soft green variety (genthite) is 42 per cent silica, 8.1 nickel oxide, 36.2 magnesia, 2.1 chromic oxide, 1.2 alumina, 1 ferric oxide, 10.8 water of crystallization.

While the quantity of ore running higher than 10 per cent nickel oxide is very small, the available quantity of ore averaging 1.75 per cent nickel oxide is enormous.

The deposit has been known since the latter part of the sixties, but being low-grade ore it has been impossible to work these properties for two reasons: first, because the former owners did not have money enough to develop a low-grade proposition of this magnitude, and second, until recently there was no process that could treat the ore without hand picking.

The nature and extent of the deposit make the mining a very simple matter, since the ore for the most part can be handled with a steam shovel and can be put into cars at a cost not exceeding 11 cents per ton.

The latest attempt to smelt this ore has been by means of the electric furnace, and as most of the nickel is used in the

steel industry, it was decided to make in one step from the ore a nickel silicide which could be sold directly to nickel steel makers.

Various samples of the nickel silicide thus produced analyzed about 10 to 25 per cent nickel, 55 to 50 iron, 30 to 20 silicon, aluminum, magnesium, chromium, etc., 5 per cent. The presence of the silicon, aluminum, and magnesium is in no way objectionable in view of the purpose for which the alloy is intended, since they act as purifiers and gas removers in steel refining.

1. An experiment was first made in which it was tried to reduce the ore directly by hot reducing gases (CO) at a high

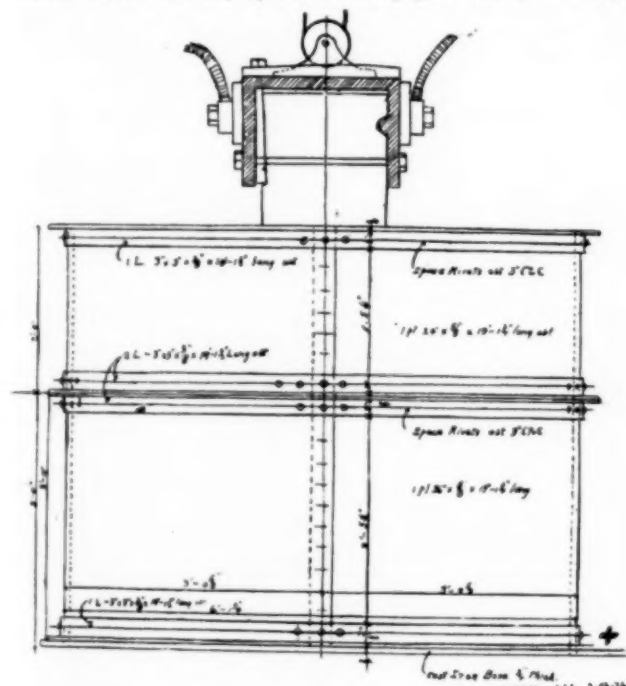


FIG. 1.—SIDE VIEW OF FURNACE.

temperature. The ore mixed with charcoal was placed in a vertical flue and the hot reducing gases passed upward through it. But although the temperature of the ore mass was raised to 1100 deg. C. and held there for an hour, there was absolutely no reduction to metallic nickel. The nickel appears to be held very strongly combined as the double silicate of magnesia and not to be reduced or affected in any way by the hot carbonous oxide gas.

2. About 60 to 75 per cent of the nickel ore runs very high in magnesia (25 to 38 per cent), but low in iron. The fact that the ore is so high in magnesia, makes it a difficult proposition to form a fluid slag, even in the electric furnace, which produces the highest temperature of any present commercial furnace. The idea naturally suggested itself, that the ore could be subjected to a high temperature, in a reducing atmosphere, and intimately mixed with charcoal, and then removed as soon as reduction fusion had taken place. The semi-fused mass could then be crushed and the nickel separated from the slag by a magnetic separator, or on the Wilfley concentrator.

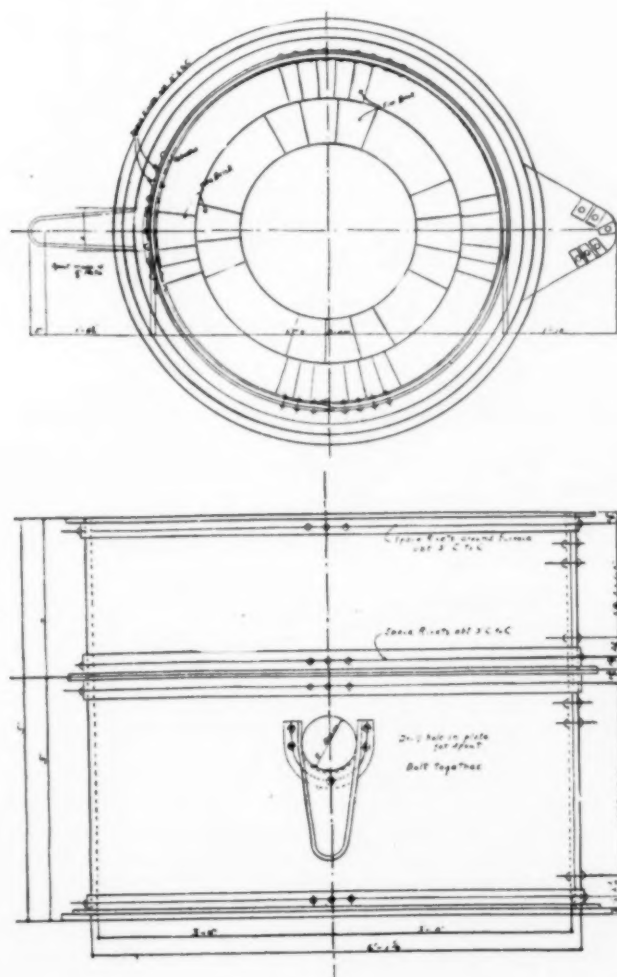
The experiment was carried out in a heating chamber heated by an electric arc, with an inclined bottom, so that the fused mass would flow off in a continuous stream. It was crushed and concentrated on the Wilfley table. The amount of ore treated per kw.-day was very low (14 pounds) compared to the results of a fusion or slag-making electric furnace, which will smelt 23 to 27 pounds of ore per kw.-day. The concentrates from the Wilfley table were high in nickel, but the total extraction of nickel from the ore was about 60 per cent, which is very low.

Further, a furnace of this kind would, after being run for a few hours become so strongly heated that the lining would be in serious danger of falling in from fusion. The experiments along this line were discontinued for they did not appear very promising.

3. Wet processes were then tried. Experiments were made with sulphuric acid, ammonium chloride, sodium chloride, sodium sulphate, and chlorine gas, but neither gave satisfactory results.

4. **Electric Smelting.**—Laboratory experiments were made with a 100-ampere 50-volt Borchers furnace, obtained from Eimer & Amend, with a fixed lower electrode and a movable upper electrode, and also with a larger furnace of the same general type.

The furnace was tapped about every 15 to 20 minutes, giving about 10 lb. (4.5 kg.) of slag and metal, about a quart (liter) in all. The author thinks that with this furnace it is not neces-



FIGS. 2 AND 3.—PLAN AND FRONT ELEVATION OF FURNACE.

sary to tap so often, first because the efficiency of the furnace is increased if the slag is removed as soon as it has become inactive in the reactions within the furnace; second, if the "source of heat" which is near the dividing line of the slag and ore be raised too much above the bottom of the furnace, the metal on the bottom will be cooled to its solidifying point, and the furnace will start to freeze. Further, if the furnace is let run too long between tapplings, carbides are formed in the slag, which renders it very infusible and sticky, so that it will not flow.

The capacity of the furnace was limited to the capacity of the gas engine, which gave a maximum of 15 kw. at the switch-

board. The furnace would treat about 100 lb. (45 kg.) of ore in six hours.

Several charges of the green and brown ores were heated with charcoal in graphite crucibles in a gas blast furnace.

The object of this special series of experiments was to get an idea of the self-fluxing qualities of the green and brown ores.

For this purpose the relative weight of brown ore in the mixture was varied from 90 to 10, the corresponding weight of green ore from 10 to 90, a weight 5 of charcoal being added in each case. A mixture of 40 weight of brown ore, 60 of green ore, and 5 of charcoal appeared to give the best slag. Since the greater part of the ore is high in magnesia (the

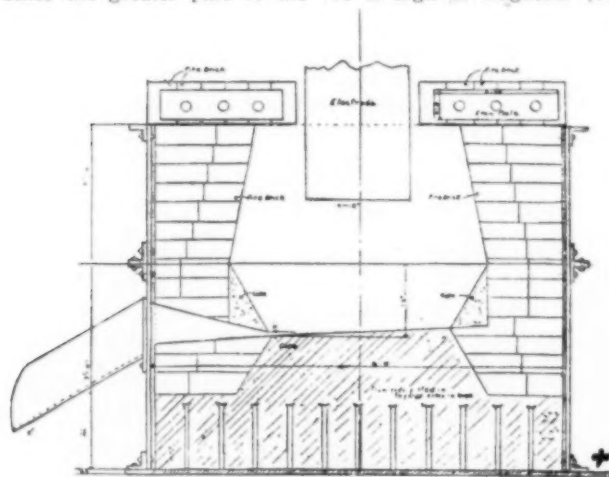


FIG. 4.—SECTION OF FURNACE WITH LINING.

green ore), it is an advantage to be able to use the brown ore as flux instead of barren matter.

Under good working conditions the slags varied about as follows:

Silica	40.00 to 45.00 per cent
Alumina	30.00 to 20.00 "
Ferric oxide.....	2.00 to 0.50 "
Calcium oxide.....	3.00 to 10.00 "
Magnesia	20.00 to 15.00 "
Nickel oxide.....	0.40 to 0.80 "
Alkalies	4.60 to 8.70 "

All the above slags gave little or no trouble in the furnace, flowed freely and allowed the metal to separate and collect in the bottom of the slag pot.

In the smelting of this ore in an open furnace there was considerable loss of nickel by volatilization. The alkalies and magnesia also volatilized to a considerable extent. The loss by volatilization is from 15 to 20 per cent of the weight of ore.

Lime in conjunction with carbon up to certain limits plays a very important part in the amount of silicon reduced. The author thinks that the formation of calcium carbide acts as a very powerful reducing agent and breaks up the silicates, liberating silicon and forming more complex silicates.

The iron aids in the collecting of the metallic nickel, very little iron goes to the slag. At first considerable trouble was experienced with the coke, it would accumulate in the furnace and finally cause it to freeze up. This trouble was practically eliminated by using finer coke, instead of $\frac{1}{2}$ -in. (1.2 cm.) coke 10-mesh (0.25 cm.) coke was used which gave better results. In order to get complete reduction and a good extraction it is necessary to add a little excess of coke, and being in the fine condition it is easily carried from the surface with the slag.

The formation of carbides in the slag is a signal that the furnace will soon give trouble, unless the charge is changed or the carbon lessened. The carbides are easily detected by the odor of acetylene about the cool slag. A slag which decomposes as it cools is another indication that something is wrong

with the fluxing. The furnace must be tapped at comparatively short intervals of not more than 30 to 40 minutes.

Granulation of the slag was tried and was quite successful; the slag is then easily removed from the metal by simply washing. The metal remains in fairly large pieces, and in the form of shot. On a large scale, granulation would be a very cheap method of separating the metal, and the slag could be easily flushed to the dump.

The Wilfley Concentrator with material through 30-mesh (0.8 cm.) gave excellent results.

The metal reduced from the ore and separated from the slag is in all cases an alloy of nickel, iron and silicon. In no run were they able to obtain a low-silicon alloy, and keep their extraction about 80 per cent. This alloy must from necessity carry considerable iron. The amount of metal separated from a ton of ore depends upon the ore treated. The brown ore will give about 440 lb. (200 kg.) of metal to the ton, while the

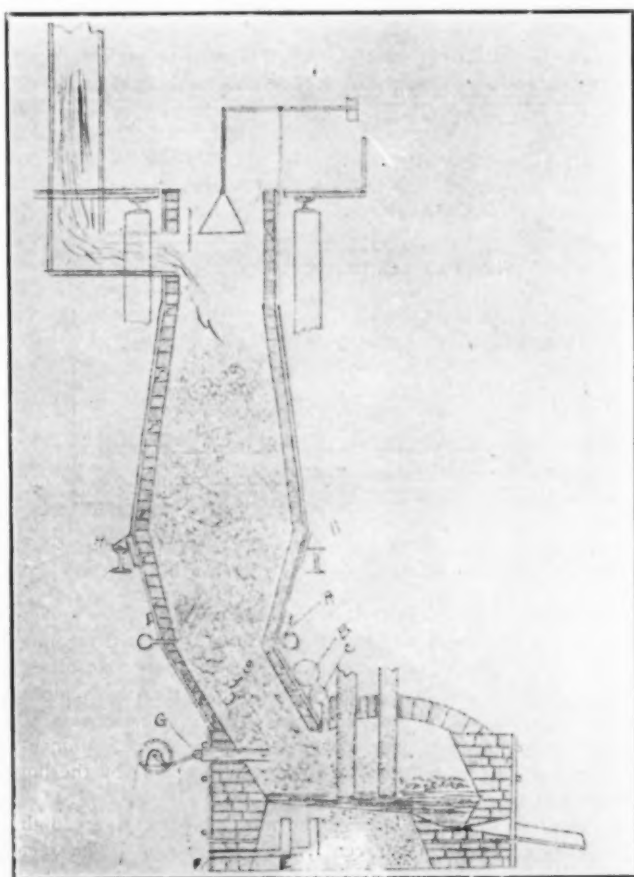


FIG. 5.—ELECTRIC SHAFT FURNACE.

green ore will not give more than 200 lb. (91 kg.) to the ton. Using a mixture of the ores the amount of metal derived, varies from 300 to 350 lb. (136 to 159 kg.) to the ton.

The following are analyses of metal obtained from the brown ores:

Nickel	10.90	11.00	11.50	10.20
Iron	58.30	56.80	57.40	56.90
Silicon	28.20	30.10	26.30	27.40

The following are analyses obtained from green ore:

Nickel	23.95	21.80	22.60	19.30	18.96
Iron	50.70	51.20	48.90	49.10	54.00
Silicon	21.20	18.80	23.24	26.40	22.80

The following are analyses of metal obtained from the mixtures of the ores:

Nickel	15.04	13.70	13.10	13.30	14.80
Iron	54.80	53.90	56.00	57.80	54.70
Silicon	26.60	27.40	25.70	24.00	26.20

In looking over the above analyses, it will be noted that the green ore gives a metal much higher in nickel, all the runs from which this metal was derived were made on picked ore averaging about 6 per cent nickel. It is not advisable to pick ore too much, as it will leave a worthless ore which it would not pay to treat; then too the amount of rich ore to the lean ore is about 1 to 10 or 20 and means a large expense to sort out.

The ferronickel silicide also contains aluminum, calcium, magnesium in traces, besides 1.5 to 2.5 per cent chromium.

Notes are given in the paper on the methods of analysis of the ferronickel silicide and the ore.

Commercial Electric Smelter.—The electric smelter and equipment is located on the nickel deposit at its middle point.

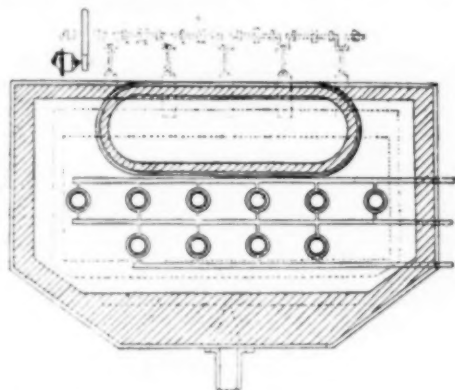


FIG. 6.—DETAILS OF FURNACE CONSTRUCTION.

In the boiler room there are four horizontal-fired tube boilers of 200 boiler hp. each, bituminous coal being fired. The steam engine is of the Hamilton Corliss type. The line shaft drives two 170-kw. generators, giving 3500 amperes at 500 volts direct current.

The furnace room equipment consists of the furnace and electrode control, switchboard, 4 slag pots, 500 lb. (225 kg.) capacity, and weighing scales for ore and flux.

The furnace used in Webster is similar to the one which was used at Sault Ste. Marie by Dr. P. L. T. Heroult, with a few modifications. The furnace was originally designed in two sections, it being intended that the top section could be removed in order to patch up the crucible, but later this was found to be unnecessary.

The bottom plate of the furnace is of cast iron 1.5 in. (3.8 cm.) in thickness, set on porcelain blocks for insulation. Tapered holes were bored in the cast-iron plate, and iron rods 10 in. (25 cm.) long by 0.5 (13 mm.) diameter were inserted, in order to give a large area of contact. A large copper plate was fastened to the cast iron at the rear, to which the lugs carrying the cables were secured.

Fig. 1 gives a side view of the furnace, the holder is shown, and method of holding the carbon electrode, also position of cables. The furnace shell is made up of 5/8-in. (16 mm.) iron plate, and 3 in. by 3 in. by 3/8 in. (7.5 x 7.5 x 0.95 cm.) angle irons.

Fig. 2 gives a plan of the furnace, showing the thickness of the fire-brick lining.

Fig. 3 is the front elevation of the furnace, showing slope of hearth and position of spout, one spout being used for both the slag and the metal.

Fig. 4 is a section of the furnace after the lining is in; also showing the covering used.

The shell of the furnace was insulated from the cast iron bottom with heavy asbestos and fibre bushings around the bolts. The interior of the shell was first lined with 1/4-in. (0.6 cm.) asbestos. A mixture of hot graphite and glucose was tamped into bottom of furnace around the iron pegs. Above the pegs the amount of graphite was lessened, and coke dust substituted, and near the top of the bed coke dust with a

linder was used. The coke lining was carried up the side for a distance of ten inches; this coke lining appears to resist the corrosive action of the slags better than any other material we used. The cover of the furnace consisted of fire brick clamped together by tie rods.

The lining of the furnace required about four days, one man and helper. The graphite and glucose was heated in an iron soap kettle. Materials used in the lining and cover of the furnace were as follows: One barrel glucose, 3/4 ton coke, 2,000 fire bricks, 1 barrel sand, 1 ton graphite, 1,000 circular fire bricks, 2 barrels fire clay, 20 sheets asbestos.

After the lining was all in, a wood fire was built in the furnace and kept burning three days. This dried the furnace very well. After this the electric current was run in the furnace for several hours, which finished the drying out of the bottom of the furnace.

Operation of Furnace.—The furnace was started with a little coke in the bottom, but after a few trials this was found to be an unnecessary precaution. The electrode was controlled by a hand-wheel located on the floor, connected to a worm and gear on a carriage, by a chain.

To the gear on the carriage was secured a notched wheel which carried a chain connected to the electrode holder. The holder is very heavy, being made of cast iron and weighing about 1,000 lb. (450 kg.). The carbon electrode which was first used weighed about 1,400 lb. (635 kg.) when new. During the operation of the furnace if the current was shut off for

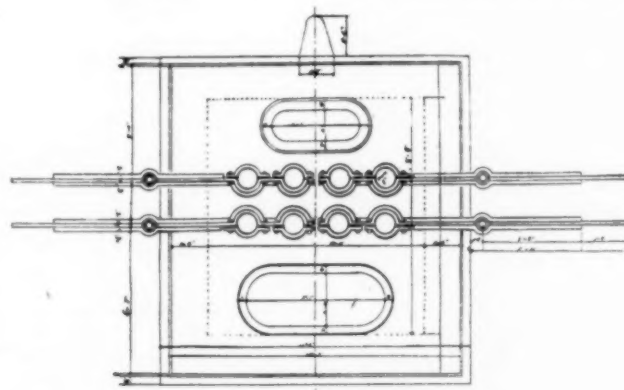


FIG. 7.—CRUCIBLE FURNACE

more than an hour it would usually cause the furnace to freeze up. At first the coke caused trouble in the furnace by accumulating, but this was eliminated by using finer coke and a little less of it. They also noted a decrease in the electrode consumption when finer coke was used.

The electrodes which we first used were of carbon, built up from smaller electrode sections. The large electrode was 18 in. (45 cm.) square by 7 ft. (2.2 m.) long, made from 3 by 6 in. (0.75 x 15 cm.) carbon slabs. The first time the electrode was heated by the arc it cracked and pieces broke off in several places. The pieces were picked out of the furnace, and ore with flux fed in, trusting that after a slag bath had been formed the cracking would stop, however this was not the case; after having to stop frequently to pick out solid pieces of the electrode weighing 25 to 50 lb. (11 to 23 kg.), the furnace was shut down, this electrode after so much damage being too short for further use. It was removed and the other carbon electrode placed in the holder.

Before striking an arc with the second carbon electrode a wood fire was placed beneath it in order to heat it up evenly. After two hours of this treatment the electrode was placed in the furnace, contact made, and ore started in. The electrode, however, went to pieces.

A copper bushing was then ordered for the holder, so that they could use two Acheson graphite electrodes together. The Acheson electrodes were spliced so as to feed continually with-

out waste. These electrodes held up finely and gave very good results. The dynamos gave trouble, however, and they could only draw half load, consequently for nearly all the runs only one 8-in. (20 cm.) Acheson graphite electrode was used, which carried as high as 5,000 amperes at times.

Trouble was also experienced with the ore if not dried in advance.

The "efficiency" of the furnace under the present condition is said not to be very high. The author says: "The efficiency of this furnace as compared with one utilizing the waste gases to preheat and dry the ore, I would estimate at 60 to 70 per cent. The fact that we used wet ore instead of dry ore makes a difference of 6 to 8 per cent in efficiency."

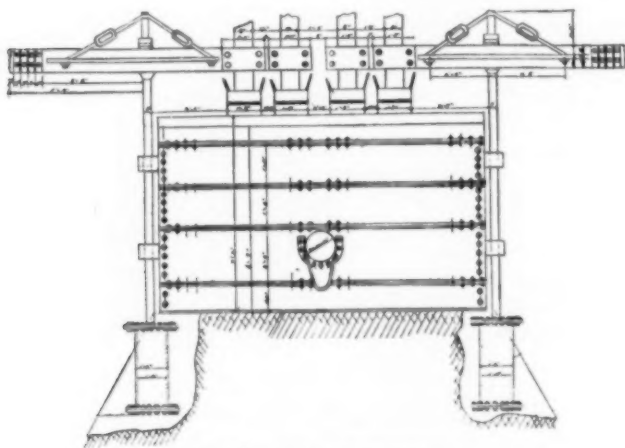


FIG. 8.—CRUCIBLE FURNACE

The following is the record of a furnace run made on December 9, 1909:

CHARGES.	
No. 1.	No. 2.
4 tons brown ore.	3 tons brown ore.
1.02 tons limestone.	0.6 tons limestone.
0.48 tons coke.	0.48 tons coke
Moisture in ore	42.50 per cent
Total dry ore treated	8,050 pounds (3,660 kg.)
Net time of smelting	45.75 hours
Limestone used	3,600 pounds (1,635 kg.)
Coke used	1,940 pounds (882 kg.)
Power used, average kilowatts	170
Kilowatt hours used	7,755
Kilowatt-years used	0.896
Ore treated per kw.-year	9,000 pounds (4,090 kg.)
Ore treated per kw.-day	24.7 pounds (11.2 kg.)
Nickel silicide obtained	1,242 pounds (565 kg.)
Per cent of nickel in alloy	11.90
Silicide obtained per kw.-year	1,380 pounds (627 kg.)
Silicide obtained per kw.-day	3.8 pounds (1.7 kg.)

In other runs, from 8,250 lb. (3,750 kg.) to 10,160 lb. (4,620 kg.) of ore was treated, and from 975 lb. (443 kg.) to 2,230 lb. (1,014 kg.) of nickel silicide obtained per kilowatt-year of electric energy consumed. The power consumption is nearly constant per unit weight of ore treated, independent of its metal contents, showing that the chief worth of the furnace is the drying of the ore and the smelting of the slag. The silicide produced contained 10.50 to 22.60 per cent of nickel.

The cost of production varies with the nickel content of the ore and of the silicide formed. The cost of smelting by hydroelectric power is only about 14 per cent of what steam power cost us. The above figures are quite conservative in the case of the hydroelectric power.

Cost of Operation.—The cost of operation is based on the approximate cost as the plant was operated from October, 1909, to April, 1910, and goes to show the expense of operating or trying to operate in a primitive country. Two shifts of 11 and

13 hours respectively were run. The furnace was run about 4 to 5 days out of every week, very little repair work being necessary on the furnace. The engine gave us considerable trouble and during every run several stops were necessary for adjustment of the crank pin or the valves. The generators giving the trouble they did, and forcing the plant to operate at 50 per cent of full load, caused a great decrease in the efficiency of the power plant. The cost of smelting cannot therefore be reasonably figured from the cost of operation which the author gives as follows:

Cost of operation per 24 hours:

1 Superintendent	\$10.00
1 Assistant	2.50
2 Oilers @ \$1.80	3.60
2 Firemen @ \$2.40	4.80
2 Assistant firemen @ \$1.80	3.60
2 Head furnacemen @ \$2.50	5.00
4 Assistant furnacemen @ \$1.80	7.20
2 Miners @ \$1.50	3.00
1 Blacksmith	2.00
1 Timekeeper	2.00
Coal, 18 tons, @ \$4.50	81.00
Machine oil	4.00
Electrodes	9.10
Flux and coke	3.00

\$148.80

The average power at the switchboard was 200 kw. and on

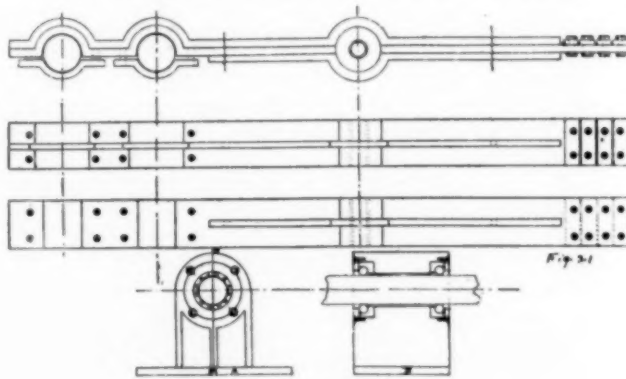


FIG. 9.—ELECTRODE CONTROL

an average 7,800 lb. (3,500 kg.) of ore were smelted per 24 hours.

Cost of power: \$0.74 per kw.-day, \$270.00 per kw.-year.

Cost of power under full load would be: \$0.562 per kw.-day, \$206.00 per kw.-year.

In the above figures no account is taken of depreciation, interest, and repairs, which would undoubtedly raise the above figures 10 per cent.

Output of Plant.—The output of the plant was limited to the available output of the generators, and varied from 800 to 1,200 lb. (365 to 545 kg.) silicide per day. This output on the present plant could be raised to 2,500 lb. (1,135 kg.) of nickel silicide, by operating the generators at full load, and using a dryer and preheater with the furnace. However, the furnace has served its purpose for a commercial demonstration, and has given the operators a chance for observations which are only possible with a furnace of this kind.

In the concluding part of this paper the author gives a discussion of various furnace designs and a proposed improved electric smelter design for treatment of the ore under discussion with estimated cost sheets.

The Shaft Furnace.—The shaft furnace for electric smelting is the ideal furnace. However, since the flow of gases is slow compared to the air blast furnace, it does not need to be so very high. For ordinary operations on the mixed nickel ores a shaft furnace should not be more than 10 to 12 feet high, i. e.,

above the fusion zone. There is considerable fines in the ore, which would cause trouble with a higher shaft. For this ore a dryer would be necessary, utilizing the waste gases from the furnace. Fig. 5 gives a general idea of the shaft furnace the author has in mind.

The ore with the flux is introduced at the top of the furnace from the charging floor. The furnace has a fore hearth for the collecting of slag and metal. The furnace is designed to carry ten 8-in. (20 cm.) graphite electrodes. The ore enters the crucible back of the electrodes. To aid in the distribution of the sticky ore mass, a set of plungers (G) are arranged at back of furnace.

The furnace is designed for direct or alternating current (single phase). The bed of the furnace acts as one conductor, the dotted portion shows this bed. Tubular conductors carry

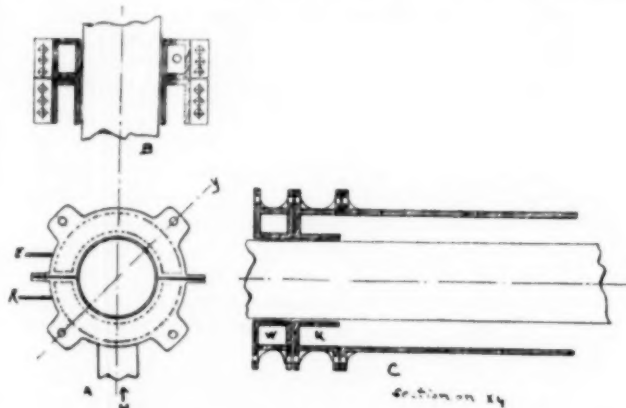


FIG. 10.—DETAILS OF JACKET CONSTRUCTION.

the current to iron distributors in the bed of furnace, which is heat insulated.

Coke is introduced at (c) independent of the charge; by so doing the electrode consumption is lessened; air enters for combustion of gases in the shaft at (A). The author has found that, by introducing the coke or reducer around or near the electrode, the consumption of electrode can be cut down one-half. Each electrode is operated automatically and independent of the others.

Although the shaft furnace is very good in theory, he does not think it practical at this time for the ore in question. The ore is extremely refractory and shut-downs are frequent. The cost of shutting down a shaft furnace and repairing would be very high. If it were desired to run the extremely fine ore alone at any time, then the crucible furnace would be the one to use. It would be impossible to run the extremely fine ore in a shaft furnace successfully. In the treatment of iron ores and other ores, the shaft furnace no doubt would be all right.

The crucible furnace.—By crucible furnace the author has reference to a furnace which is comparatively shallow and has no shaft for preheating and reducing the ores.

The crucible furnace illustrated in Figs. 7 and 8 is of a special design, which the author sketched, after careful observations on the present furnace. The furnace as shown is designed for direct current, which at the present time is installed.

The furnace is designed to carry eight 8-in. (20 cm.) Acheson graphite electrodes, operating under a maximum load of 2,000 electric h. p. The electrodes are worked in pairs and controlled by automatic hydraulic regulating device. The electrodes are carried in cast steel holders, to which are fastened eight flexible cables.

The holders are supported on a three-inch steel rod, which works in a hydraulic ram Fig. 8. To give the rod a free movement a special guide was designed, shown in Fig. 9. The ball bearings will not allow the rod to hang if properly lubricated and kept covered.

Fig. 7 gives a front view of the furnace, showing position of spout and electrodes. Fig. 8 is a side view of the furnace and gives an idea as to method of making the lower contact. The conductors are heavy copper tubing which enter the lower bed of the furnace; to these are secured wrought iron distributors.

The carbon bottom of furnace is not brought to the outside wall of furnace, but to the brick lining, which cuts down the radiation losses. There are openings in the top of furnace for electrodes, and a flue for escaping gases and an opening for ore.

The gases are taken off from the front part of the furnace and burned in the preheater. The ore as it leaves the preheater is dumped into the furnace at the rear. The two flues are brick lined. The interior of the furnace is lined very much in the way the experimental furnace was lined.

The electrodes are water jacketed; below the water jacket is a jacket constructed for the introduction of coke around the electrode. In Fig. 10 is a plan of jacket, B and C are sections. W shows section of water jacket. The water enters at E and flows out at R. K represents section of coke jacket. The coke enters at M and is fed by a screw device.

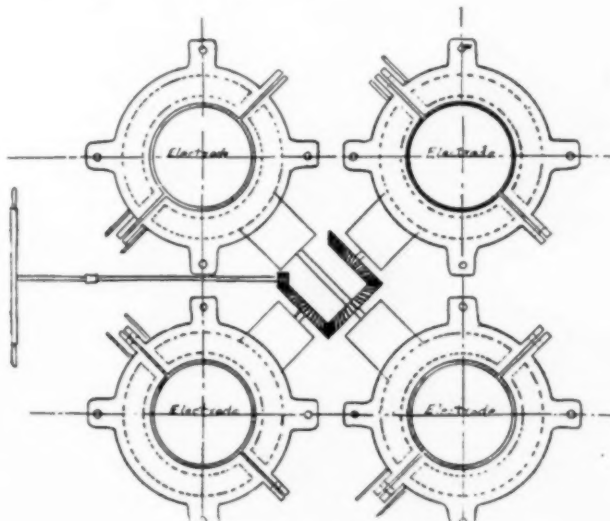


FIG. 11.—INTRODUCING COKE TO ELECTRODES.

The general plan of introducing coke to electrodes is shown in Fig. 11, feeding coke to four electrodes, the other four being taken care of in the same way. The two jackets are suspended on tie rods, then built in top of furnace. They are built in two sections so they may be easily renewed. For the first operations it is intended to feed the coke intermittently by hand feed.

By introducing the coke at this point, an extremely high reducing atmosphere is constantly kept around the electrode, consequently the electrode consumption is cut down. As to the reducing power, or action on the ore by this method, it is about the same.

The fusion zone is limited to a few inches immediately around the tip of the electrode, extending perhaps 8 or 10 inches (20 to 25 cm.). Under normal conditions there is a violent bubbling and boiling, which causes the fine coke to 20 mesh (2.5 to 5 mm.) to intimately mix with the fused mass and bring about reduction. On a small scale the author has found that this method of introducing the coke works out very nicely. With frequent tappings the excess of coke is carried away in slag.

Electric smelter design.—Fig. 12 shows a proposed electric smelter design for the treatment of this ore. It consists of the generator room, furnace room, and ore bins.

The generator room contains the motor-generators for the

changing of the current from three phase to single phase. The room contains two 3,000 kw. motor-generators, the exciter, the high-tension switchboards, transformers and recording-meters. Current is to be furnished by outside power stations.

In the furnace room there are the furnaces, dryers, switchboards, slag and metal conveyors, hydraulic accumulator, and pumps. It is intended to keep three furnaces in operation, and have one for emergencies, which can be used at any time.

All electric cables are carried in conduits, except the cables to the electric holders. The hydraulic control is so arranged with each furnace that it may be regulated by hand or automatically.

All metal and slag is tapped into an iron trough of water, and is carried to the concentrating room by endless conveyor, where slag is flushed to dump.

The dryer is of such length that the ore is dried and pre-heated before entering the furnace. A check draft and ex-

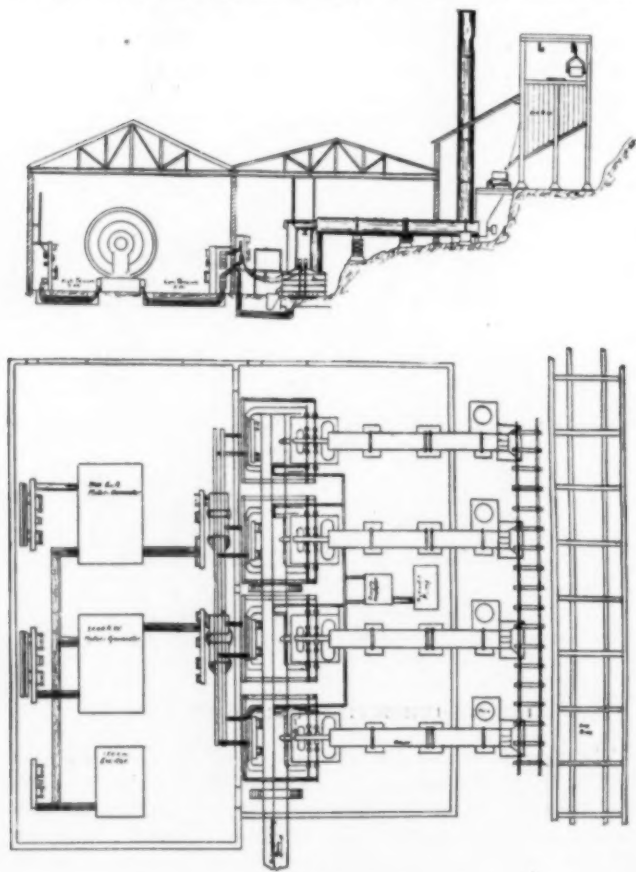


FIG. 12.—ELECTRIC SMELTER DESIGN.

plosion door is arranged at end of dryer. Each dryer has a stack to give better regulation.

The limestone and ore are crushed before being placed in the bins. From the bins they are drawn into weigh cars, then dumped into the feeding hopper of the dryer. A screw feed regulates the amount of ore being fed into the dryer.

Estimated cost of 100-ton plant.—Approximate cost of a 100-ton electric smelter, including aerial tramway, rock house, crusher, cars, buckets, etc.:

Ore bins.....	\$ 3,000
Buildings	12,240
Tramways, cars and equipment.....	25,000
Weigh car.....	1,000
Crusher	900
Pulverizer for coke.....	800
Motors	1,600
Furnaces (5).....	44,000
Motor-generators	45,000

Transformers	20,000
Structural work.....	22,000
Dryers and motors.....	15,000
Switchboard and equipment.....	30,000
Erection, labor, materials, etc.....	60,000

Total\$280,540

Although these figures appear quite high, from past experience the author thinks them conservative.

Cost of operation.—Estimate as to cost of operation of a 100-ton smelter, figuring operating 300 days per year, and cost of power \$25.00 per kw.-yr.

Expense per day:

Interest and depreciation (14%).....	\$130.00
Mining @ 15 cents per ton.....	15.00
Power	450.00
Labor and supervising.....	75.00
Electrodes	130.00
Repairs	10.00
Coke and limestone.....	75.00

Total cost.....\$885.00

From 100 tons of ore on an average 15 tons of ferro-nickel-silicide would be extracted.

$\$885.00 \div 15 = \59.00 per ton, average cost of metal alloy.

The author concludes that the present plant for commercial operations is out of the question; that the wet process now existing is also out of the question, and that the only process available for this ore at present is the electric furnace process in a first-class efficient plant.

Entertainment of Ladies.

The program of the entertainment of ladies included a reception on Thursday afternoon by Lieut.-Col. and Mrs. J. B. Miller, at their residence. This was greatly enjoyed.

On Thursday evening there was a theater party at the Royal Alexandra ("Rebellion" with Miss Gertrude Elliott in the leading part.)

On Friday morning a motor drive was made through the city to the Lambdon Golf Club for lunch.

This report should not be concluded without emphasizing again the splendid hospitality extended to the Society by Toronto. The thanks of the visitors are due to all Toronto members and local committees, but special credit should be given to the chairman and secretary of the general committee, Dr. W. Lash Miller and Mr. W. P. Cohoe.

In the following we give an alphabetical list of all members and guests who registered at the meeting:

T. B. Allen, Niagara Falls, N. Y.; J. E. Babb, Pittsburgh, Pa.; L. H. Baekeland, Yonkers, N. Y.; T. F. Bailey, Alliance, Ohio; J. W. Bain, Toronto; W. D. Bancroft, Ithaca, N. Y.; E. A. Beck, New York; P. McN. Bennie, Niagara Falls, N. Y.; S. L. Bigelow, Ann Arbor, Mich.; Wm. Brady, Chicago, Ill.; Wm. C. Bray, Boston, Mass.; J. W. Brown, Cleveland, Ohio; C. L. Bryden, Scranton, Pa.; R. H. Bulley, Syracuse, N. Y.; A. Burton, Toronto; J. E. Callahan, Niagara Falls, N. Y.; H. R. Carveth, Niagara Falls, N. Y.; A. P. Cohoe, Toronto; E. R. Cole, Niagara Falls, N. Y.; W. M. Corse, Buffalo, N. Y.; I. L. Crider, Cleveland, Ohio; R. A. Crider, Cleveland, Ohio; Wm. C. Cuntz, New York; C. Dantsogen, Scotia, N. Y.; Tos. Davies, Toronto; S. Dushman, Toronto; J. W. Evans, Belleville; Mr. and Mrs. F. A. J. Fitzgerald, Niagara Falls, N. Y.; M. W. Franklin, Schenectady, N. Y.; R. H. Gaines, New York; A. E. Gibbs, Wyandotte, Mich.; Miss M. E. Gibbs, Wyandotte, Mich.; Thos. W. Gibson, Toronto; W. L. Goodwin, Kingston, Ont.; A. M. Hamblet, Rumford, Me.; C. A. Hansen, Schenectady, N. Y.; H. E. T. Haultain, Toronto; Carl Hering, Philadelphia, Pa.; P. Heroult, New York; A. Hirsch, Corpus Christi, Texas; B. H. Hite, Morgantown, W. Va.; A. H. Hooker, Niagara Falls, N. Y.; L. E. Howard, Lockport, N. Y.; W. B. A. Howk, Elyria, Ohio; T. C. Irving, Jr., Toronto; A. van Isakovics, Monticello, N. Y.; O. S. James, Toronto, Ont.; A. J. Jones, Niagara Falls, N. Y.; W. P. Kaufmann, Toronto; J. Kelleher, Toronto; Mrs. and Miss Kelleher, Copenhagen; P. H. Kemble, Toronto; P. T. Kerivan, Toronto, Ont.; E. C. Klipstin, New York; Wm. Koehler, Cleveland, Ohio; F. L. Koethen, Niagara Falls, N. Y.; M. M. Kohn, New York; A. T. Laing, Toronto; J. Langmuir, Schenectady, N. Y.; W. S. Lecky, Toronto; F. A. Lidbury, Niagara Falls, N. Y.; C. F. Lindsay, Bridgeport, Conn.; M. G. Lloyd, Chicago, Ill.; Thos. R. Loudon, Toronto, Ont.; J. Lundgren, Chicago, Ill.; K. S. MacLellan, Toronto; J. H. Mason, Toronto; W. Lash Miller, Toronto; J. W. Moffat, Toronto; G. W. Morden, Agric. College, Winnipeg; W. F. Morrison, Lockport, N. Y.; J. M. Muir, New York; Dr. and Mrs. E. F. Northrup, Princeton, N. J.; A. Nueghorn, Toronto; C. H. Ohlweiler, Altoona, Pa.; A. C. Parsons, Buffalo, N. Y.; Chas. L. Parsons, Durham, N. H.; N. K. B. Patch, Toronto; M. Petinet, Niagara Falls, N. Y.; Jos. Race, Toronto, Ont.; M. Randall, Poplar Bluffs, Mo.; C. H. Rich, Clanton, Pa.; Jos. W. Richards, So. Bethlehem, Pa.; T. D. Robertson, Toronto, Ont.; Mr. and Mrs. A. G. Rodgers, Niagara Falls, N. Y.; E. F. Roeber, New York; S. S. Sadtler, Philadelphia, Pa.

L. E. Saunders, Niagara Falls, N. Y.; C. N. Schluederberg, Pittsburgh, Pa.; W. B. Schulte, Madison, Wis.; J. A. Seede, Schenectady, N. Y.; Louis Simpson, Ottawa, Ont.; C. V. Slocum, Pittsburgh, Pa.; A. Smith, Niagara Falls, N. Y.; W. Snow, Penn. Ill.; R. C. Snowden, Niagara Falls, N. Y.; J. L. K. Snyder, Cleveland, Ohio; T. Somery, Lyons (France); E. C. Sprague, Niagara Falls, N. Y.; T. A. Stamps, Niagara Falls, N. Y.; W. C. Stone, Watertown, Wis.; Mr. and Mrs. E. R. Taylor, Penn. Yan, N. Y.; R. W. Thornson, Toronto; R. Turnbull, Welland; Mr. and Mrs. L. D. Vorce, Detroit, Mich.; G. B. Waterhouse, Buffalo, N. Y.; Dr. and Mrs. O. P. Watts, Madison, Wis.; R. J. Weitlaner, Buffalo, N. Y.; W. R. Whitney, Schenectady, N. Y.; Roy Winne, Schenectady, N. Y.

Screenless Diametric Sizing of Ore.

A prerequisite for successful dry concentration of ore is close sizing. This holds true for separation based on the principle of difference of specific gravity and for electromagnetic or electrostatic separation of raw or roasted ores. The necessity for close and accurate sizing increases with the complexity of an ore; that is, when several minerals of narrow differences in specific gravity are found in the same ore.

Quite naturally screens of various forms and methods of operation have been used for diametric sizing and such devices are widely used for separating coarser grades. But the difficulties of diametric sizing increase with the fineness of the pulp and consequently hydraulic classification is more commonly used on the finer sizes. Nevertheless it would be preferable in some cases to continue sizing into the finer grades, obtaining sized products within narrow limits without respect to the specific gravity of the constituent particles.

A screenless sizer is not easily conceived of, but such is the nature of the machine invented by Messrs. C. L. McKesson and B. F. Rice, of Colorado Springs, Col. Their dry concentrator was described in our issue of September, 1910, page 549, and it was for the purpose of obtaining accurately sized products for this machine that the screenless sizer was evolved.

The work of the sizer bears no relation to that of a concentrating table, but for the purpose of getting a mental picture of it a comparison can be made. On first sight the sizer gives the impression of a table deck set at an angle of almost 45°, vibrating somewhat after the manner of a concentrator and fed with ore at about the same relative position. This is as far as the comparison can go, however, for the principle of specific gravity does not enter into the work of the sizer as with a table.

By reference to the accompanying drawing and photograph the following description will be understood. The working surface of the sizer is a deck 15 ft. long, 6 ft. wide at the back or feed end and 4.5 ft. wide at the head or discharge end. The deck is heavily trussed to prevent vibration and mounted at an angle of 38° on four inclined toggle supports which, in turn, are firmly supported on a base of I-beams. The deck is attached to a head movement which imparts a longitudinal vibration and its weight is counterbalanced on four compression springs, two under the upper and two under the lower side.

The working surface of the sizer consists of a number of so-called deflectors extending across its width. These deflector surfaces are about 6 in. wide and rise successively from the deck at an angle of about 9°, giving the appearance of a series of steps from one end of the deck to the other. They are not arranged in a uniformly parallel manner on the deck, but their side lines correspond to slightly diverging radii extending from an imaginary center above and behind the deck. Thus the deflectors on either half of the deck have different angles of inclination, corresponding to the positions which would be assumed by a pendulum swinging through a gradually decreasing arc.

The deflectors are covered with corrugated rubber, the line of corrugations extending lengthwise of the deck, or, more exactly, corresponding to numerous arcs described from the imaginary center referred to above. The corrugations are about 1/8 in. deep and number eight to the inch.

In operation the speed of the head movement is 320 r.p.m. The motion imparted to the deck is a combination of the reciprocating head movement, modified by the toggles. On its forward movement the deck is also raised vertically by the

toggles and lowered on the return movement. The accompanying Fig. 3 represents graphically the two motions. One marked "stroke" represents the horizontal motion, and the other marked "toggle" shows the vertical throw.

The material to be sized is fed at the upper corner of the

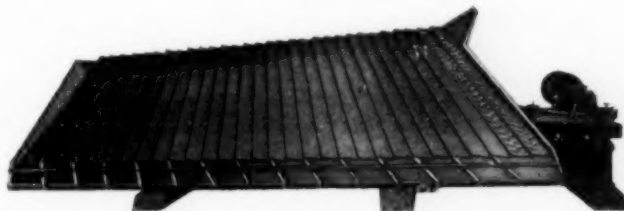


FIG. 1.—WORKING SURFACE OF SCREENLESS SIZER.

deck nearest the head motion. As the particles fall onto the deck they are acted upon by a combination of forces, including the peculiar motion of the machine and the arrangement of deflectors and corrugations, and make their progress across and down the deck. The coarsest particles will not progress beyond the first two deflectors which are covered with metal baffles, as shown in Fig. 1, and which act like a scalping screen

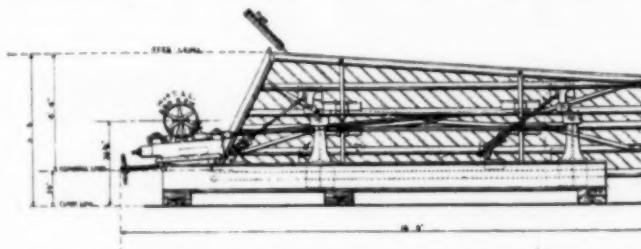


FIG. 2.—REAR VIEW OF SCREENLESS SIZER.

in cutting out an oversize for regrinding. Particles of successively smaller size will progress across and down the deck until they reach a critical point, beyond which they will not pass, whereupon they roll down the deflector and off the deck. The finest particles and dust remain on the deck longest and come off at the last deflectors or over the end of the deck. It

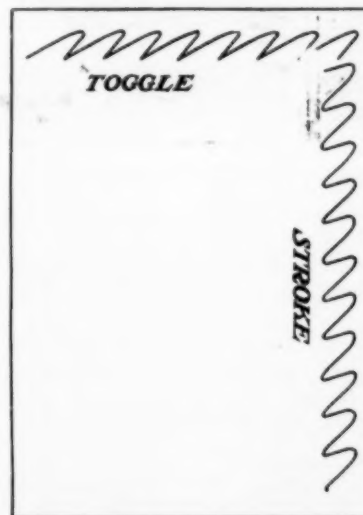


FIG. 3.—DIAGRAM OF DECK MOTIONS.

must be remembered that this separation is wholly with regard to size and without respect to the specific gravity of the particles. Along the bottom of the deck is arranged a series of cutting vanes to aid in collecting the various products according to grade. These cutters are adjustable, even when the machine is running. It is found that the zones yielding different grades do not necessarily correspond with a deflector, or any number of them, so the cutters must be set in positions determined by grading analyses to give the desired products. With the adjustments of the sizer once properly made it will continuously give uniform grades. The length of the stroke and inclination of the toggles will vary with requirements, and the capacity of the sizer also will depend in a measure on these adjustments as well as on the size of the material being treated. Recent tests indicate a capacity of from 15 to 20 tons per day of 24 hours on material crushed to about 20 mesh. It is ap-

parent, however, that a double deck can be arranged like an inverted V, which would have double the capacity, be lighter and more rigid, and at the same time require not much more power than the single deck, which is about 1 hp. The life of the sizer should be about the same as that of a standard concentrating table, and the cost of maintenance should be no more. The soft rubber surface of the deflectors should resist the abrasion of the ore very well.

The sizing efficiency of the machine is high and equally good work can be done on the fine and coarse sizes. The following tables show screen analyses made on 15 products obtained in a test. The ore was ground to pass a 24-mesh screen. The first table shows the uniform grading of the various products when divided into plus and minus 50 and 100-mesh portions. The second table gives screen analyses of the different products. It will be noticed that the grading efficiency in some of

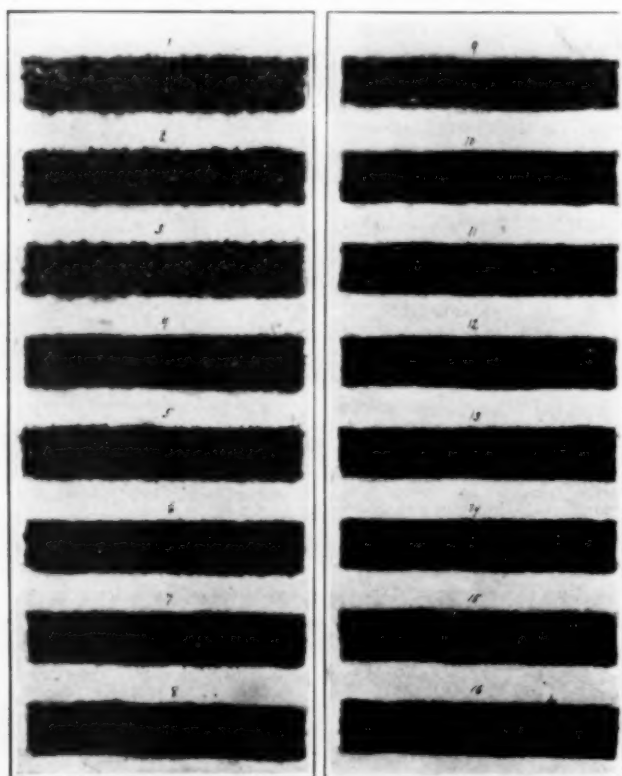


FIG. 4.—CARD SHOWING THE DIFFERENT SIZES MADE.

the products is lower than in others, but it is claimed that this can be remedied by adjusting the cutters to throw the oversize and undersize into the preceding and succeeding grades respectively, and thereby increase the efficiency of the sizing in all grades.

In addition to its use on ore, the sizer can find application in other industrial lines where sizing is required. Among the many materials requiring grading are sand and gravel, glass sand, abrasives, phosphate rock, mineral paints, salt, explosives, grains and seeds.

TABLE I.

Material ground through 24-mesh screen; graded into 15 products, each being screened through 50 and 100-mesh screens.

Product No.	Grade.	Screen.	%	Screen.	%
1	Coarse	+50	100	-50
2	"	+50	100	-50
3	"	+50	98.44	-50	1.56
4	"	+50	87.50	-50	12.50
5	"	+50	67.18	-50	32.82
6	Medium	+50	6.25	-50	93.75
7	"	+50	1.00	-50	99.00
8	"	+100	99.00	-100	1.00
9	"	+100	93.75	-100	6.25
10	"	+100	81.25	-100	18.75
11	"	+100	75.00	-100	25.00
12	"	+100	50.00	-100	50.00
13	Fine	+100	32.00	-100	68.00
14	"	+100	25.00	-100	75.00
15	"	+100	6.25	-100	93.75

TABLE II.

Grading analyses on 15 products of the McKesson Sizer. Ore originally ground through 24-mesh screen.

Product No.	Screens.	Per Cent.	Efficiency %.
1	-24 +30 +40 -40 15.60 81.20 3.20	81.20
2	-28 +40 -40 93.75 6.25	93.75
3	-30 +34 +40 +44 -44 10.92 50.00 32.83 6.25	82.83
4	-34 +40 +44 +50 -50 28.13 37.50 25.00 9.37	62.50
5	-40 +44 +50 +66 -66 7.82 25.00 64.06 3.12	89.06
6	-44 +50 +66 +82 -82 6.25 68.75 21.87 3.12	90.52
7	-50 +66 +82 -82 37.50 53.13 9.37	90.63
8	-50 +66 +82 +97 -97 9.37 62.50 25.00 3.13	87.50
9	-50 +66 +82 +97 -97 6.25 50.00 37.50 6.25	87.50
10	-66 +82 +97 -97 37.50 46.87 15.63	84.37
11	-66 +82 +97 +116 -116 25.00 50.00 21.87 3.13	75.00
12	-66 +82 +97 +116 -116 15.63 43.75 34.37 6.25	78.12
13	-66 +82 +97 +116 -116 6.25 37.50 43.75 12.50	81.25
14	-82 +97 +116 +139 -139 25.00 43.75 28.12 3.13	71.87
15	-82 +97 +116 +139 +172 -172 6.25 18.75 43.75 25.00 6.25	68.75
Average.....			81.32

Automatic Temperature Control.

The object of the temperature controller described in this article is to maintain the temperature of a liquid, a vapor or a gas automatically and exactly at a desired point. It is made by the C. J. Tagliabue Manufacturing Company, of 396 Broadway, New York City.

The action of this temperature control is based on the fact that different materials have different coefficients of expansion with temperature. If two strips of different metals, but of equal length, are placed side by side and welded together at one end only, then on raising or lowering the temperature the lengths of the two strips will generally change, but not in the

same degree, and the difference in the change of the length of the two strips can be made use of.

This difference in the change of lengths is used to actuate a valve mechanism operated by compressed air.

Fig. 1 shows the general scheme for a case in which a tank of water is to be maintained at a uniform temperature. If left to itself it would cool off, but it is kept at the desired temperature by injecting steam. The amount of steam which is admitted at any moment is regulated so as to keep the temperature constant; and the mechanism of the valve which regulates the supply of steam is operated by compressed air, and the air valve is actuated by the two-metallic expansion stems, as will be described below.

A is the water tank, *B* is the pipe for the steam which heats the water. *C* is the valve in this pipe which regulates the amount of steam which is passed at any moment into the water. This valve is actuated by compressed air from the supply line *D*. The air-valve mechanism which regulates the pressure of

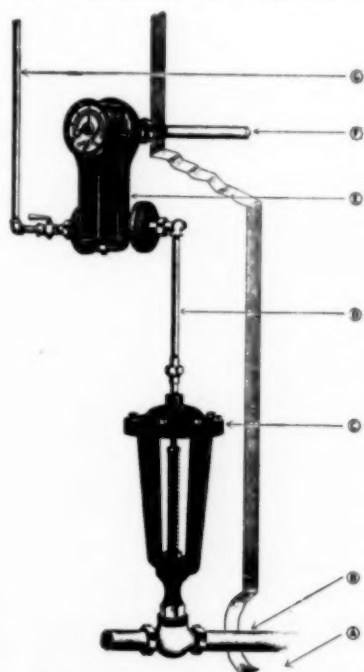


FIG. 1.—TEMPERATURE CONTROLLER.

the air for the valve is shown at *E*, while *F* is the expansion stem of the controller, which expands or contracts more or less according to the temperature of the water in the tank. It supplies the movement for operating the valve mechanism of the controller. *G* is the pipe line which carries the initial supply of compressed air which is required as auxiliary motive force.

In case the temperature of the water would for some reason rise slightly beyond the desired temperature on which the controller is set, then the expansion stem *F* of the controller will expand, and its slight movement will be greatly multiplied by the lever mechanism of the controller and will then be transmitted to the air valve mechanism which now will admit more compressed air to the valve, so that the valve will now admit less steam to the water tank, and so on.

Fig. 2 shows the interior mechanism of the controller with the expansion stem. *J* is the inner rod of the expansion stem, *K* the outer tube part of the stem. The inner rod *J* is made of a material of zero coefficient of expansion; that is, the length of the inner rod remains always the same, whatever the temperature. On the other hand, the outer tube part *K* is made of a material with a high coefficient of expansion so that its length changes considerably with a change of temperature.

The controller is attached to the tank by means of the

threaded hub *L* in such a way that the stem *K* extends unobstructedly into the apparatus.

The left-hand end of the inner rod *J* of the stem is constantly pressed by means of the spring *M* against the plug at the left-hand end of the tube *K*; hence when the outer tube *K* elongates or contracts then the inner stem *J* (which remains of constant length) is forced by the spring to move in unison.

To the right-hand end of the inner stem *J* is attached an extension rod *N* which is made of the same material as the expanding outer tube *K*. Hence, the inner and outer parts of the expansion stem to the right of the hub *L* are contracting and expanding equally, so that any changes in temperature outside of the apparatus, the temperature of which is to be controlled, do not affect the functioning of the controller.

It is, therefore, only the inner and outer parts of the expansion stem to the left* of the hub *L* which are made of

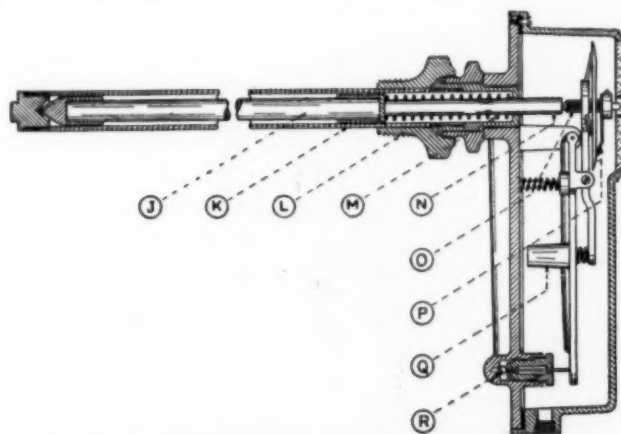


FIG. 2.—EXPANSION STEM AND CONTROLLER MECHANISM.

different materials and which are, therefore, affected unequally by a variation of temperature.

The extension *N* is held against the point of the adjusting screw *O*, the adjustment of which determines at which temperature the controller will function. To the adjusting screw *O* is attached the pointer *P* which indicates the adjustment on a dial beneath, facilitating close adjustments.

Q is the lever mechanism through the short end of which the adjusting screw *O* passes. At this point the lever is held by spring tension against the extension *N*.

The auxiliary motive force (in this case compressed air) is controlled by the valve mechanism *R*. This valve has three ports; one for the admission of the motive force (compressed air), another for supplying the diaphragm motor and another for the escape of the diaphragm-motor supply. The construction is shown on a larger scale in Fig. 3.

U is the inlet port through which the initial supply of motive force (compressed air) enters from the compressed-air supply line; the connections for the line being shown at *T*.

V is the valve-ball, which assumes a position in accordance with the position of the valve stem *X* against which the long end of the lever *Q* in Fig. 2 is held by spring tension, thus regulating the amount of compressed air passing through the ports.

W is the outlet-port connecting with *Y*, the outlet (if the position of the valve-ball *V* allows).

Y is the connection for the line carrying the motive force between the controller and the diaphragm motor.

Z is the waste-port, which provides an escape for the motive force (compressed air) when it is in excess of the amount needed to hold the diaphragm-motor valve open or shut to the desired extent.

*That is the parts within the apparatus the temperature of which is to be maintained constant.

We can now trace in detail, by means of Figs. 2 and 3, what happens if the water in our tank (which is to be held at constant temperature) shows a tendency to rise in temperature.

The temperature will rise, indeed, very slightly. But this will at once result in an elongation of the outer tube *K* of the expansion stem in Fig. 2. As *K* is fixed to the tank at the connection *L*, the left-hand end of *K* will move away from *L* to the left. The inner rod *J* is not affected in its length by the change of temperature and is therefore caused by the spring *M* to move along to the left. The extension *N* moves in unison and consequently the upper arm of the lever *O* moves also, on account of the spring tension which keeps *O* constantly in touch with *N*.

Since the one arm of the lever *O* moves one way (to the left in Fig. 2), the other arm moves in the opposite direction (to

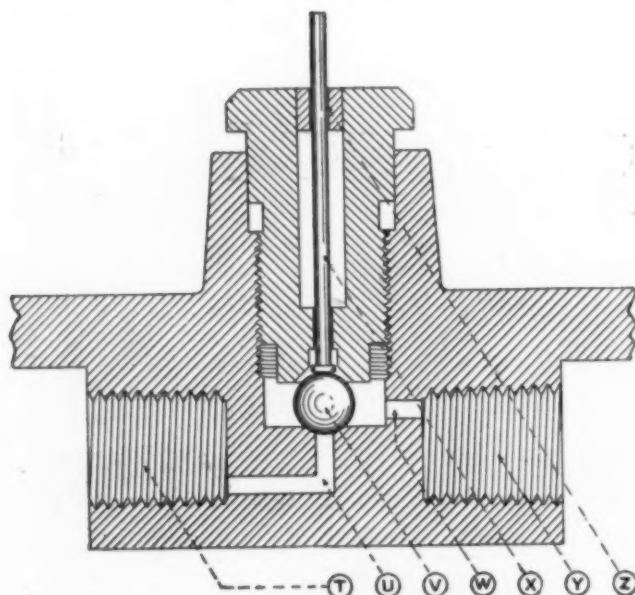


FIG. 3.—VALVE MECHANISM.

the right in Fig. 2), a pivot being arranged between the two ends of the lever.

Consequently the valve stem *X* will now move also (upward in Fig. 3). The valve-ball *V* being pressed constantly against the head of *X* by the compressed air supply will now also move upward so that more air can enter through the inlet *U* and pass through the port *W* and connections *Y* to the diaphragm motor. Thus a greater pressure of air is supplied to the diaphragm motor, which thereby closes the steam valve to a greater extent and curtails the steam supply to the tank.

Should, however, the water in the tank show a tendency to cool off so that more steam is required, the controller will cause the diaphragm-motor valve to open further (thus letting in more steam) by functioning in just the opposite manner from that above described.

In practice the diaphragm-motor valve is rarely wide open or dead shut. Usually it is "throttling"—the controller being so sensitive to the slightest change in temperature within the apparatus to which the controller is attached that the diaphragm-motor valve is constantly "throttling"—letting in just a little more or just a little less steam, as required to keep the temperature constant.

The above-described design and operation permit of various modifications. Compressed air was assumed as the motive power and is generally best suited for this purpose. The initial supply should be of at least 15 lb. per square inch pressure, but the volume needed is comparatively slight.

The controller may also be built in some cases, however, for operation by water (instead of compressed air) as motive power. But when water is used the construction of the valve

mechanism is somewhat different, although the principle is the same.

In the above example the controller mechanism was so arranged that when the stem of the instrument is subjected to a higher temperature the diaphragm motor closes the valve which controls the heat supply. This is the "direct-acting" type of controller.

In other cases in practice it may, however, be required that a rise in temperature be checked by turning on a greater supply of a cooling medium. In such cases the "reverse-acting" type of controller is used. In this type the controller mechanism is so arranged that when the controller stem is subjected to a higher temperature the diaphragm motor opens the valve further.

The many applications of this controller for industrial work, especially in the chemical industries, are so evident that only a few typical applications may be mentioned.

In the case of air to be heated by steam coils to a given temperature the controller stem is placed in the air duct (in which the temperature is to be adjusted) and the diaphragm-motor valve adjusts the steam supply to the steam coils.

If the temperature of a heated air in a closed chamber (an incubator, dryer, etc.) is to be maintained constant the controller may be used with both diaphragm-motor valve and diaphragm-motor lever. In case the temperature tends to rise the diaphragm-motor valve decreases the steam supply for the heating coils, while the diaphragm-motor lever opens the damper for the air supply to a greater extent.

For this controller, which is one of the latest Roesch-designed devices, the following principal points of advantage are claimed:

The inner rod of the stem of the controller is of a material which has a zero coefficient of expansion. If the inner stem has even a slight coefficient of expansion (though much less than the outer portion) the action of the controller would not be changed in principle, but in practice the controller would show the troublesome phenomenon of "creeping" due to the fact that the functioning point of the controller varies as the temperature effect slowly permeates the internal part of the stem. It would, therefore, take such a controller quite a while to settle down. With the above-described controller, with its internal stem remaining rigidly of constant length, all this trouble is avoided.

There is but one valve mechanism in the working parts of this controller, and this valve mechanism is also self-cleansing and self-adjusting. Since the proper working of the device does not depend upon a number of parts being in fine adjustment in relation to each other the effects of wear and tear are greatly reduced.

The cover which incloses the working parts of the controller can be removed while the controller is in use, and thus the working parts be exposed while in operation, thus permitting easy inspection.

No stuffing box arrangements are used in the mechanism. Further, that part of the mechanism which multiplies and transfers the movement of the expansion stem to the valve mechanism is never in contact with the compressed air supply.

Automatic Temperature Control in Electric Ovens.

For use in research as well as industrial laboratories the easy and exact temperature control which is possible with electric furnaces is one of their greatest advantages. Usually this temperature control is obtained by some sort of rheostat which is adjusted by hand.

For many purposes it would, however, be greatly advantageous to have automatic temperature control, but the problem in general meets with great difficulties. For the restricted field of electrically-heated ovens with temperatures from that of air to 175° C. (347° F.) the Freas oven, which has just been placed on the market by Messrs. Eimer & Amend, of New

York City, has solved the problem. Even in the restricted field, indicated by the above given temperature limits, the exact automatic temperature control should prove exceedingly useful for many purposes.

The temperature-regulating device in the Freas oven is entirely of metal and its action depends on the expansion of a metallic tube running through the chamber from top to bottom and operating a lever which makes and breaks a contact, proper means being provided to prevent arcing. The lever is extended to serve as an indicator operating up and down a temperature scale on the outside of the oven near its top. By turning the screw at the bottom of this temperature scale the indicator is moved to the desired temperature; this is all that is required of the operator and the regulating device does the rest in maintaining the temperature to which the furnace is set.

The oven, as shown in the adjoining illustration, is built of asbestos wood, which has the triple advantage of being fire-proof, a good heat insulator, and not attacked by acids or alkalis. The oven is fitted with one or two shelves, perforated with small holes to distribute the heat uniformly

high accuracy the oven is equipped with an extra fan, driven by a motor.

To heat the furnace up to 50° C., about 15 minutes' time is required with maximum current, and 45 minutes to heat it up to 110° C.

A Liquid Flow Recorder.

Liquid flow recorders have become a necessity in recent years chiefly for two kinds of work; firstly in large power stations for the measurement of condensed steam and boiler feed-water, and secondly in chemical and metallurgical works for recording the rate of flow and the quantities of acids and solutions in operation. There can be no doubt that especially in the latter field their use will rapidly increase, when the practical reliability of modern flow recorders is properly understood. In the following we give a description of the Lea liquid flow recorder, which was first developed in England and is placed on the market in this country by the Yarnall-Waring Company, 1109 Locust Street, Philadelphia, Pa.

Liquid flow recorders are essentially based on the laws governing the flow of water over wires and notches. The Lea flow recorder employs for this purpose a sharp-edged V notch to which the formula of Prof. James Thompson applies, according to which the flow in cubic feet per minute through a 90-deg. notch is $0.305H^{3/2}VH$, where H is the height of the notch in inches.

For simplicity's sake, let us assume that water is the liquid, the flow of which is to be measured. Then it will be easy to plot a "curve of flow"; that is, a diagram which gives the rate of flow of water for different depths H of the water according to Thompson's formula. If now a float is placed in the water, the rise and fall of this float may be connected to an indicating or recording device, such as a recording pen, through an intermediary mechanism in such a manner that the recording device moves at any instant in direct proportion to the rate of flow of the water in that instant, and therefore plots on a revolving drum a curve which is directly a continuous record of the flow of water during any length of time. Clearly the problem is to devise a suitable and exact intermediary mechanism, as mentioned above. This problem is solved in the Lea recorder in an ingenious manner in the following way:

The float in the water rises and falls with the water level. Above it is the instrument case, and the float spindle passes through the bottom of the instrument case upwards into the latter. Upon this float spindle a rack is cut, which gears into a small pinion upon the axis of a drum, revolving between centers.

Upon the body of the drum is a screw thread, the contour of which is the "curve of flow" (as defined above) for the V notch in connection with which the recorder is used, and just as the flow through the notch increases rapidly with the depth, so the pitch of the screw increases in the correct proportion.

Above this drum is a horizontal slider bar, supported on small pivoted rollers, and carrying an arm, at the upper end of which is a pen or pencil point in contact with a paper chart, upon a clock-driven recording drum, which revolves once in twenty-four hours.

As the float rises or falls, the drum spiral is rotated, and its motion is imparted to the slider bar (and pen arm), by means of a saddle arm, projecting from the latter and engaging at its lower end with the screw thread.

On the body of the spiral drum is a graduated scale, adjacent to the screw thread, and computed in pounds per hour. It is thus seen that the spiral drum serves a double purpose:

Firstly, its screw thread rectifies the motion of the pen, so that it moves equal distances for equal increments in the rate of flow.

Secondly, it provides a magnified scale for making an accurate observation of the rate of flow at any moment.



ELECTRIC OVEN, WITH AUTOMATIC TEMPERATURE CONTROL.

throughout the chamber. This insures uniform temperature, there being less than one degree difference between the top and bottom.

The door is fitted with a mica window, so that parts of the chamber can be inspected without opening the door. Complete ventilation to get rid of fumes and moisture from the substance being dried is obtained by two openings on each side of the oven and two through the top.

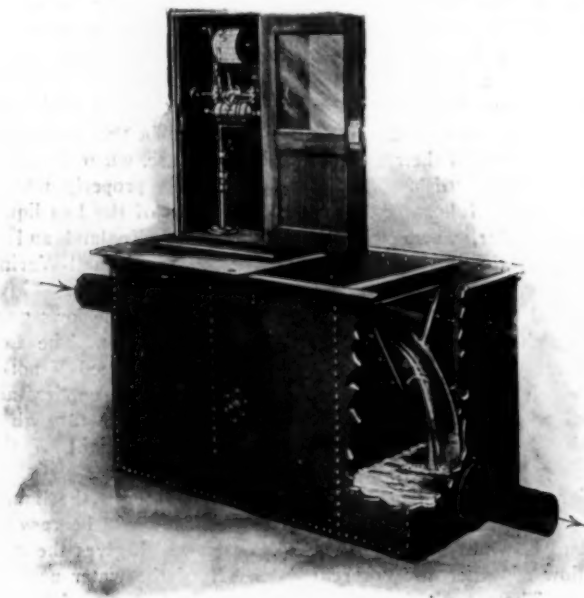
The heating resistors are placed at the bottom of the oven and consist of wire coils so insulated that they are not injured by acid fumes, etc.

The furnace can be operated on alternating current as well as on direct-current circuits. The power consumption is 660 watts, which for the lower temperatures can be reduced one-half by means of a switch, provided the closest regulation is desired.

The regulator at any temperature between air and 175° C. is sharp and accurate to less than one degree and remains so indefinitely, while under most favorable conditions it is accurate to one-tenth degree. For continuous operation at such

Thus the actual depth of water in the notch can be observed at any time. The instantaneous rate of flow, in pounds per minute or gallons per minute, can be read off directly. Finally the recording pen which moves in direct proportion to the rate of flow plots a diagram whose area is a measure of the total quantity of liquid passed.

The adjoining illustration shows the lower part of the apparatus in which the water acting on the float passes through



LIQUID FLOW RECORDER.

the V notch, and the upper instrument case with the vertical float spindle and the record drum. The illustration is otherwise self-explanatory.

When the flow to be measured is so small that it would show only a depth of 1 or 2 in. in a 90-deg. V notch, notches having angles less than 90 deg. may be used, like 45-deg., 22.5 deg., etc. Speaking generally, the flow through such notches is approximately proportional to their width, for a given depth.

Hence by means of these acute notches, a large float movement is obtained when only a small quantity is being measured.

Further, a measuring tank and recorder having a maximum capacity of say, 10,000 gal. per hour through a 90-deg. notch, if supplied with two other notch plates—say 1 of 45 deg. and 1 of 22.5 deg.—can be used for measuring either 10,000, 5000 or 2500 gal. per hour.

Moreover, one recorder may be used for measuring large or small quantities of liquids by providing several identical notches, of which a greater or smaller number may be closed according to the requirements. With such a multiple-notch arrangement the total flow at any instant is obtained by multiplying the reading on the recorder by the number of notches open.

It is clear that in principle the operation of the apparatus is exactly the same whether it is used for measuring the flow of water or of acids or any other liquids. But for the use with corrosive liquids it is all-important that nothing but the float is in contact with the liquid, the instrument case itself being above the flow arrangement, out of contact with the liquid.

In the case of a sulphuric acid solution, for instance, the lower tank through which the acid passes, may be made of lead-lined wood, and the notch also of lead. But glass may be used equally well.

Another point which is important for certain chemical operations, is that a float may be placed above the weir to control the inlet, thereby insuring a constant head of the liquid. This

is especially valuable in manufacturing processes where a definite percentage of solution has to be added.

There seems no doubt that these liquid flow recorders and adjusters can be of considerable service in large-scale chemical operation.

Carboy Rocker.

The carboy rocker, shown in the adjoining illustrations, is so handy that a brief description of its construction and method of use should be interesting to all users or bottlers of acids, ammonia, etc., from carboys. It is made by Messrs. Rockhill & Vietor, 114 John Street, New York City.



FIG. 1.—ATTACHING ROCKER TO CARBOY.

This carboy rocker is attached to a carboy or detached from it in one-fourth of a minute. One man can roll a full carboy to any desired position on the floor and completely empty it without spattering. In use the carboy can be left resting on the first or second stop, as there is no need of placing it back. It is ready for immediate use.

Fig. 1 shows the method of attaching the rocker to the

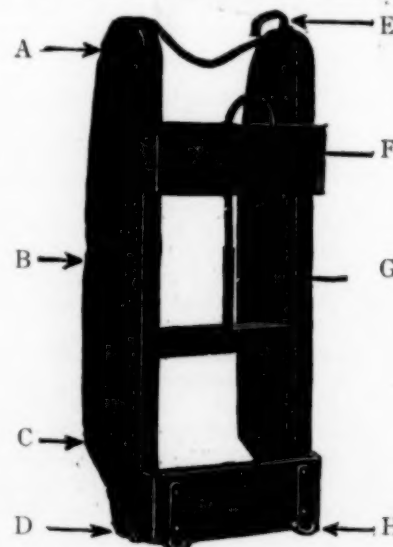


FIG. 2.—STOP CARBOY ROCKER AND VENT TUBE, DETACHED.

carboy. The workman holds the rocker with the right hand by the curved, pointed, sliding hook. He then, with his left

hand, takes hold of the cleat on the side of the carboy, lifts one side of the carboy sufficiently to catch the pointed supporting hooks underneath it, pulls the curved sliding hook up and over and presses it down on top of the rocker. He then removes the stopper of the carboy and inserts the celluloid vent tube, closing the outer end of the tube with the forefinger while inserting it.

Fig. 2 shows the construction of the rocker and vent tube detached: *a* is the last position; this point is in contact with the floor when the carboy is empty; *b* is the middle stop; this point is in contact with the floor when the carboy is half empty; *c* is the first stop; the carboy remains stationary when the stop touches the floor.

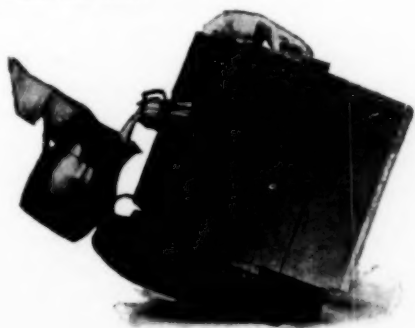


FIG. 3.—FIRST PITCHER FULL.

The wheels *d* facilitate moving. The celluloid vent tube, ready to insert in carboy, is shown at *e*; attention should be called to the fact that it is made of celluloid; *f* is a pointed steel sliding hook; *g* a spiral steel spring, and *h* are supporting hooks.

Fig. 3 shows the position of rocker and carboy when the first pitcherful is taken.

Notes.

Erratum.—The article of Prof. J. W. Richards in our September issue (p. 463) mentions 250 million tons of Lake Superior ore running 35 to 50 per cent of iron; this should have been 250 billion tons, i. e. 250,000 millions.

The **Scientific Materials Company**, of Pittsburgh, Pa., have recently enlarged their glass-blowing department, increased their force, and added the latest and approved machinery. This department is now in charge of an expert German glass blower, who has made a specialty of intricate laboratory apparatus. An expert engraver takes charge of the graduating of the most precise volumetric apparatus. Special glass apparatus according to drawings may be obtained on short notice, and broken apparatus will be repaired and returned promptly.

The **Zaremba Company**, manufacturers of evaporators and chemical machinery, with main offices in Buffalo, have opened a branch office in New York, Room 95, 97 Liberty Street.

The **Brown Instrument Company**, the first concern to manufacture pyrometers in the United States and whose office and laboratory have been located at 311 Walnut Street, Philadelphia, since 1860, has been forced to move to larger quarters by the great increase in its business. As electrical instruments play an important part in the construction of pyrometers, it was found desirable to be able to superintend the construction of milli-voltmeters for use in their pyrometers. With this end in view, Mr. Brown, the president of the Brown Instrument Company, has now also become president of the **Keystone Electrical Instrument Company**, of Philadelphia, which, for a number of years, has been manufacturing voltmeters and ammeters, and the shops of the Brown Instrument Company are being combined with those of the Keystone Electrical Instrument Company at Ninth Street and Montgomery Avenue, Philadelphia. This places the Brown Instrument Company in the unique position of controlling the construction of their milli-voltmeters for their pyrometers, even to the manufacture

of the raw materials, such as the magnets. By this arrangement the Brown Instrument Company and the Keystone Electrical Instrument Company will manufacture pyrometers, thermometers, speed indicators, draft gages, recording gages, voltmeters and ammeters. In fact, they are now in a position to manufacture or repair practically any type of instrument. In addition to the works and office at Ninth Street and Montgomery Avenue, Philadelphia, an office will also be maintained for the present at 311 Walnut Street, and the branch offices of the Brown Instrument Company at Pittsburgh and Chicago will also be continued.

Carbon Electrode Factory at Niagara Falls.—The new carbon plant which the National Carbon Company, of Cleveland, Ohio, is building at Niagara Falls will be ready for occupancy some time in October. The company has been working for about a year in building this new branch. It will be a thoroughly modern factory in every respect and will engage in the manufacture of carbon electrodes exclusively. This is the second plant which the National Carbon Company has built during the past year, the other being at Toronto. In nearly every one of their nine factories considerable improvement and expansion has been made during this year. The main plant and general offices of the National Carbon Company are located at Cleveland. This plant alone covers about 30 acres of ground, and is engaged in the manufacture of a miscellaneous line of carbon products. The eight outside plants manufacture a special line of product for which the locality, equipment and organization is best adapted. The Cleveland plant is by far the largest industrial carbon factory in the world, while the total facilities of the nine branches make a tremendous organization for the development and manufacture of a complete line of high-class carbon products.

The production of secondary metals, meaning those recovered from scrap metal, drosses, sweepings, etc., amounted to a considerable industry in the United States in 1910, according to figures compiled by the U. S. Geological Survey. Of secondary copper, including that in alloys other than brass, there were produced 25,396 short tons; remelted brass, 27,347 tons; secondary lead, 17,822 tons; recovered lead in alloys, 23,327 tons; secondary spelter, 33,040 tons; recovered zinc in alloys other than brass, 1,181 tons; secondary tin, 2,423 tons; recovered tin in alloys, 3,092 tons; secondary antimony, 27 tons; recovered antimony in alloys, 1,529 tons. The total value of these metals and products was \$45,525,500. The production of secondary copper amounted to 17 per cent. of the domestic consumption of new copper. The secondary lead was equal to 11.5 per cent. of the refined lead produced in the United States. The secondary zinc including that in brass equaled 23.2 per cent. of the total production as primary spelter in the United States. The secondary antimony and tin constituted practically the only domestic production of those metals.

Quicksilver in California.—In 1910 California produced 83.5 per cent. of the total output of that metal in the United States. The county leading in production is San Bernardino in which are the New Idria mines, at present the most important producers in the American hemisphere. Fifteen quicksilver mines contributed to the California output in 1910, which amounted to 17,211 flasks of 75 lb. each, valued at \$800,484.

The Bureau of Mines has issued within the past month two technical papers; No. 3, Specifications for the Purchase of Fuel Oil for the Government, with directions for sampling oil and natural gas, and No. 4, the Electrical Section of the Bureau of Mines. Miners' Circular No. 4 describes the use and care of mine-rescue breathing apparatus.

PERSONAL

Mr. Louis D. Bastian, of Silverton, Colo., has been engaged as superintendent of the Vermillion mill near Animas Forks, Colo.

Mr. Raymond C. Benner, of the department of chemistry

at the University of Arizona, has gone to the University of Pittsburgh, where he will investigate the smoke problem with Professor Robert K. Duncan.

Mr. Gelasio Caetani, of San Francisco, has been spending some time in the Telluride district, Colorado, making tests and advertising improvements in the milling operations there.

Mr. R. S. Davis, chemist for the Primos Chemical Company, Newmire, Colo., has returned to his work after a short leave of absence.

Dr. Wilbur A. Hendryx, of New York, has gone to South Africa on business for the Hendryx Cyanide Machinery Company.

Dr. H. R. Low, of Platteville, Wis., recently spent several weeks in the San Juan district, Colorado, in the interests of the American Zinc Ore Separating Company.

Mr. A. J. McMillan, formerly managing director of the Le Roi Mining Company, and now closing up the affairs of that company, has returned to Rossland, B. C., after spending three months in England.

Mr. Charles L. Parsons, secretary of the American Chemical Society, and until recently professor of chemistry in the New Hampshire College, Durham, N. H., has been appointed chief chemist of the mineral division of the Bureau of Mines at Washington.

Prof. Robert H. Richards, of Boston, spent the month of September in Denver.

Mr. John E. Rothwell, of Denver, has just completed a tour of the principal western metallurgical centers.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore-Treatment (Continued).

399,209, March 5, 1889, Julio H. Rae, of San Francisco, California.

Gold and silver. Treats ore held in suspension by an agitator, the metal being deposited on the cathode. Several cells are shown, each consisting of a tub of wood lined near the bottom by a vertical cathode ring of copper, silver or silver-plated metal. An agitator is journaled on a vertical shaft extending through the bottom of the cell, having radial arms carrying stirrers, some of which are inclined outward to scrape the ore from the cathode ring and throw it to the center, while others are inclined outward. The anodes consist of a number of vertical carbon rods, arranged in two concentric circles and depending from a wood ring into the upper part of the ore-pulp. Each rod has a spring connection and is readily removable.

459,023, Sept. 8, 1891, Curt Schreiber and Hans Knutsen, of Broken Hill, New South Wales.

Gold, silver and antimony. Antimony ore is hand-picked into three classes containing more than 25 per cent of antimony, 5 to 25, and less than 5. Each lot is then crushed to pass a sieve of 900 meshes to the square inch, and is lixiviated with a solution containing sodium monosulfid, in the proportion of 235 parts to each 336 parts of antimony sulfid. Or barium or calcium sulfid may be used in molecular proportions. One cu. yd. of solution, say 200 gal., is sufficient for each ton of ore, the strength being varied according to the percentage of antimony therein. The three lyes from the three classes of ore are differently treated. Each lot of ore is leached with three or more solutions, first with solution used twice previously, then with solution used once, and finally with fresh solution. Strong solution is mixed with the weaker

ones, and antimony sulfid is dissolved in the weakest ones. The resulting saturated solutions of antimony double salts are electrolyzed, the conductivity being increased by the addition of a small amount of sodium chlorid. The current is regulated according to the concentration of the lye. Antimony is deposited at the cathode, with production of sodium hydrosulfid which is oxidized at the anode to sodium disulfid. Sodium hydroxid is added from time to time to convert the sodium disulfid into monosulfid, which is returned to the lixiviators. The residual ore, freed from antimony, is amalgamated for the recovery of the gold or silver.

460,354, Sept. 29, 1891, Werner von Siemens, of Berlin, Germany, Assignor to Siemens & Halske, of same place.

Copper, etc. Various cells for carrying out the process of his patent No. 416,576. The first cell is a rectangular wooden pan, covered inside and out with lead, and having a perforated false bottom on which is supported the anode, of corrugated sheet-lead or corrugated and perforated plates of retort-carbon. Lying on the anode is a screen consisting of two spaced-apart sheets of fine wire-gauze, to prevent agitation of the liquid at the anode. The cathode consists of a number of horizontal cylinders of wood coated with wax and then with metal, submerged in the electrolyte and slowly rotated. The solution, containing a salt of the metal to be deposited, a ferrous salt and free acid, is supplied at the top of the cell, and passes from the cathode, where two-thirds of its metallic contents are deposited, down to the anode where the ferrous salts are reoxidized, being drawn off at the bottom through a syphon. The second cell consists of a semicircular or V-shaped trough, its inner surface lined with an anode having small transverse grooves. The cathode is a revoluble cylinder, as before, separated from the anode by the double wire-gauze screen. The solution enters at the top and the resulting ferrous salts are reoxidized, as before, oxidation beginning at the edges of the grooves and being completed at their bottom. The third cell is a circular pan of hard lead, on the bottom of which are supported a number of short vertical carbon rods, or a grate of horizontal carbon rods, constituting the anode. Or the anode may consist of grooved carbon plates. Above the anode is a sheet of wire-gauze, and above this is a horizontal rotating disk cathode of metal. The solution passes downward as before. The motion of the cathodes is necessary for the production of good copper and the use of a high current density.

473,105, April 19, 1892, George J. Atkins, of London, England.

Gold, silver, etc. The first cell illustrated comprises two vertical cylindrical portions, subdivided by a diaphragm of matting, flannel, felt or a porous plate, mounted to slide vertically in grooves. In and nearly filling the anode chamber is an anode comprising a shaft, e.g., of wood or protected metal, carrying a broad-bladed screw of carbon or lead. The cathode compartment contains a revolving cylinder of perforated sheet-copper. Flexible rubber scrapers project from the walls of the compartment against the surface of the cathode. The cell is filled with a solution of sodium chlorid, sodium sulfate, potassium cyanide, or potassium ferro-cyanide. Ore is fed into the top of the anode compartment and descends downward in contact with the screw. The screw may be rotated to accelerate or retard the descent of ore, the contact being sufficient to effect the desired oxidation or chlorination. The solution is re-pumped from the bottom of the anode compartment to the top. The ore, freed from base metals, generally silver and sometimes gold, is withdrawn from the bottom of the anode compartment. If the electrolyte liberates chlorine or other solvent of gold, this is generally reprecipitated by a base metal in solution, such as iron. The metals loosely deposited on the cathode are scraped off and removed below. The gold is recovered by treating the purified ore in an amalgamator. A modified cell consists of a vertical copper cylinder, forming the cathode, containing and separated from the anode screw by a diaphragm of canvas, flannel or felt.